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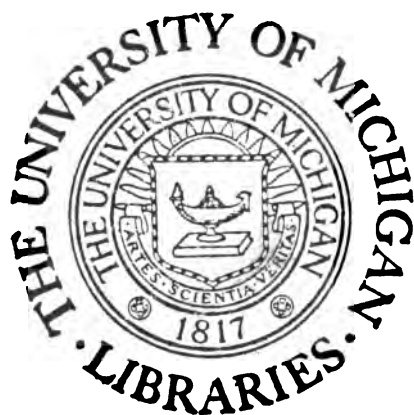
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Bulletin 67

**DEPARTMENT OF THE INTERIOR
BUREAU OF MINES**

JOSEPH A. HOLMES, DIRECTOR

**ELECTRIC FURNACES
FOR
MAKING IRON AND STEEL**

BY

**DORSEY A. LYON
AND
ROBERT M. KEENEY**



**WASHINGTON
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CONTENTS.

PART I.—THE ELECTRIC FURNACE IN PIG-IRON MANUFACTURE.

	Page.
Introduction.....	7
Development of the electric furnace for smelting iron.....	7
Investigations of the Canadian commission.....	7
Investigation of 1904.....	7
Experiments with Keller furnace, Livet, France.....	8
Problems not solved by commission of 1904.....	9
Experiments at Sault Ste. Marie, Ontario, 1908.....	10
Development of the electric iron-reduction furnace in Sweden.....	11
Experiments of Grönwall, Linblad, and Stalhane, Domnarfvet, Sweden.....	12
Experiments of the Jern-Kontoret, Trollhättan, Sweden.....	14
Description of furnace.....	15
Changes after the first experiments.....	17
Summary of working results and analyses.....	20
Report on operation of the furnace from August, 1911, to May, 1912.....	22
Durability of the roof of the crucible.....	26
Commercial furnaces of the Swedish type.....	27
Chemistry of the reduction of iron ores in the electric furnace.....	28
Problems in the electric smelting of iron ores.....	31
Early difficulties.....	31
Electrode problem.....	32
Unsolved problems.....	33
Volume of shaft as compared with capacity of furnace.....	33
Reduction of ore in shaft.....	33
Gas circulation.....	35
Method of utilization.....	36
Objections.....	36
Other methods suggested.....	37
Status of the iron industry in the Western States.....	39
Iron-ore deposits on the Pacific coast.....	39
Development of the electric reduction furnace in California.....	41
Ore deposit of Shasta Iron Co.....	41
First experimental electric reduction furnace.....	42
Second and third experimental furnaces.....	42
Present type of furnace.....	44
The plant.....	45
The furnace.....	45
The electrodes.....	45
The transformers.....	46
Electric control.....	46
Operation.....	46
Use of charcoal as a reducing agent.....	47
Use of crude petroleum as a reducing agent.....	48
Comparison of the California and Swedish furnaces and processes.....	48
Reduction in shaft.....	48
Circulation of gases.....	49
Use of calcined limestone.....	50
Electric furnace as compared to the blast furnace.....	50
Hypothetical case.....	51
Cost of power.....	52
Use of electric-furnace pig iron in the open-hearth furnace.....	52

PART I.—THE ELECTRIC FURNACE IN PIG-IRON MANUFACTURE—Contd.		Page.
An estimate for the erection of an electric iron-smelting and steel-refining plant		54
Assumptions and conditions		54
Proposed system of working		55
Items entering into estimated cost of construction		55
Cost of production		57
 PART II. THE ELECTRIC FURNACE IN STEEL MANUFACTURE.		
Introduction		58
History of the development of the electric steel furnace		58
Early development of the electric furnace		58
Early development of the electric steel furnace		59
Development of the Héroult steel furnace		59
Experiments of Stassano		60
The Kjellin furnace		62
Investigations of the Canadian commission of 1904		63
Experiments with Kjellin furnace, Gysinge, Sweden		63
Experiments with the Héroult furnace, La Praz, France		64
Conclusions of the commission		65
Later development of the electric steel furnace		66
The Girod arc furnace		66
Other arc furnaces		66
The Röchling-Rodenhauser furnace		68
Other induction furnaces		69
General lines of development of electric steel furnaces		69
Present status of the electric steel industry		71
Electric steel furnaces		74
Arc furnaces		74
The Héroult furnace		74
Single-phase Héroult furnace		74
Three-phase Héroult furnace		75
The Girod furnace		77
Single-phase Girod furnace		77
Three-phase Girod furnace		79
The Stassano furnace		81
The Keller furnace		82
The Grönwall furnace		82
The Nathusius furnace		83
Other electric steel furnaces of the arc type		84
Induction furnaces		84
The Kjellin furnace		84
The Röchling-Rodenhauser furnace		85
Other induction steel furnaces		87
An electrical resistance furnace adaptable to steel manufacture		87
Electric steel manufacturing practice		89
Manufacture of steel from scrap iron and steel in the electric furnace		89
Société Électro-Métallurgique Française, La Praz, Savoye, France		89
Description of plant		89
Practice at plant		89
Final form of product		90
Plant of Messrs. Vickers (Ltd.), Sheffield, England		91
Description of plant		91
Practice at plant		92
Lakes & Elliot foundry, Braintree, England		94
Description of plant		94
Practice at plant		94
Products		96
Girod steel plant, Ugine, France		96
Location		96
Power supply		96
Description of plant		97
Refining practice		98
Products of the furnace		100

PART II.—THE ELECTRIC FURNACE IN STEEL MANUFACTURE—Contd.

	Page.
Electric steel manufacturing practice—Continued.	
Manufacture of steel from scrap iron and steel in the electric furnace—Continued.	
Foundry of Mönkemöller & Co., Bonn, Germany-----	102
Description of plant-----	102
Foundry practice-----	102
Sheffield Annealing Works, Sheffield, England-----	102
Description of plant-----	103
Plant practice-----	103
Plant of Crucible Steel Casting Co., Lansdowne, Pa-----	104
Superrefining of molten steel in the electric furnace-----	105
Plant of Illinois Steel Co., South Chicago, Ill-----	105
Description of plant-----	105
Plant practice-----	105
Products-----	107
Plant of American Steel & Wire Co., Worcester, Mass-----	108
Gutehoffnungshütte works, Oberhausen, Germany-----	109
Description of plant-----	109
Refining practice-----	109
Products-----	116
Works at Völklingen, Germany-----	117
Method of refining furnaces-----	117
Refining practice-----	117
Products-----	118
Plant of La Gailais Metz & Co., Dommeldingen, Luxemburg-----	119
Characteristics of operation of electric steel furnaces-----	119
Design-----	119
Power consumption-----	120
Power factor-----	121
Electrodes-----	123
The lining-----	123
Essential features of refining in an electric furnace-----	124
Cost of producing steel in the electric furnace-----	125
Cost of producing steel in the Girod furnace-----	125
Cost of producing steel in the Grönwall furnace, Sheffield, England-----	126
Cost of producing steel in Röchling-Rodenhauser furnaces-----	126
Properties of electric-furnace steel-----	128
Acknowledgments-----	130
Suggestions for a duplex process for making steel-----	131
Duplex process for larger furnace-----	132
Cost of operating an electric furnace alone-----	133
Selected bibliography-----	134
Publications on mining and mineral technology-----	137
Index-----	139

ILLUSTRATIONS.

	Page.
FIGURE 1. Sectional plan and elevation of Keller furnace-----	8
2. Experimental furnace used at Sault Ste. Marie, Ontario-----	10
3. First type of shaft furnace used in experiments of Grönwall, Lindblad, and Stalhane-----	12
4. Second type of shaft furnace used by Grönwall, Lindblad, and Stalhane-----	12
5. Third type of furnace used by Grönwall, Lindblad, and Stalhane-----	13
6. First type of furnace used at Domnarfvet, Sweden-----	14
7. Second type of furnace used at Domnarfvet, Sweden-----	14

	Page.
FIGURE 8. Plan of furnace house at Trollhättan, Sweden -----	15
9. Plan of furnace at Trollhättan-----	16
10. Sectional elevation of furnace at Trollhättan-----	17
11. Sectional elevation of furnace at Trollhättan, showing arrange- ment of gas circulation-----	18
12. Diagram of electrical connections of furnace at Trollhättan--	19
13. Bus-bar connections and arrangement of electrodes in Swedish type of electric shaft furnace-----	27
14. Curves showing variation of fuel required and of heat evolved--	34
15. Curves showing variation of power required according to the composition of the waste gases-----	35
16. Héroult 1,500-kilowatt, three-phase reduction furnace-----	42
17. Second 1,500-kilowatt, three-phase reduction furnace built at Héroult, Cal-----	43
18. Sectional elevation of 2.5-ton, single-phase Héroult steel fur- nace, La Praz, France-----	60
19. Elevation of 1-ton, three-phase Stassano steel furnace, Bonn, Germany-----	61
20. Plan of 1-ton, three-phase Stassano steel furnace, Bonn, Ger- many-----	62
21. Elevation of 1.5-ton, single-phase Kjellin steel furnace, Gysinge, Sweden-----	63
22. Elevation of 2.5-ton, single-phase Girod steel furnace, showing arrangement of conductors-----	66
23. Elevation of 8-ton, three-phase Keller steel furnace, Unieux, France-----	67
24. Elevation of 2.5-ton, two-phase Grünwall steel furnace, Shef- field, England-----	67
25. Longitudinal elevation of 2.5-ton, two-phase Grünwall steel furnace, Sheffield, England-----	68
26. Elevation of Nathuasius three-phase steel furnace, showing principle of operation-----	69
27. Plan and elevation of 2-ton, single-phase Röchling-Roden- hauser steel furnace, Volklingen, Germany-----	69
28. Plan and elevation of 2.5 to 3 ton, single-phase Girod steel fur- nace, Ugine, France-----	78
29. Section of water-cooled steel electrode used in Girod furnace--	79
30. Plan and elevation of 10 to 12.5 ton, three-phase Girod steel furnace, Ugine, France-----	80
31. Plan and elevation of Hering resistance furnace-----	87
32. Arrangement of electric-furnace steel plant, Oberhausen, Ger- many-----	110
33. Results of analyses of 19 samples of metal during one heat----	111
34. Power fluctuations of 2.5-ton, single-phase Girod steel furnace during a heat-----	114
35. Specific-energy consumption as function of weight of charge. Girod furnace-----	114
36. Voltage, output, and power consumption for 2.5-ton, single- phase Girod steel furnace-----	115

ELECTRIC FURNACES FOR MAKING IRON AND STEEL.

By **DORSEY A. LYON** and **ROBERT M. KEENEY.**

THE ELECTRIC FURNACE IN PIG-IRON MANUFACTURE.

By **DAVID A. LYON.**

INTRODUCTION.

In the inquiries and investigations that the Bureau of Mines is making with a view to increasing safety, efficiency, and economic development in the metallurgical industries, the application of electricity to various processes, and especially to those in the manufacture of iron and steel, is being given attention. Some results of the work already done are presented in this bulletin, which gives a historical review of the development of electric furnaces for making iron and steel, and discusses the problems that remain to be solved in the use of electric furnaces for the smelting of iron ores and the production of pig iron at a profit on a commercial scale.

DEVELOPMENT OF THE ELECTRIC FURNACE FOR SMELTING IRON.

In 1898 Capt. Stassano, of Italy, patented an electric furnace for smelting iron ores.^a In 1900 the production of ferro-alloys in the electric furnace was begun, and at the present time practically all ferro-alloys are made in electrically heated furnaces.

INVESTIGATIONS OF THE CANADIAN COMMISSION.

INVESTIGATION OF 1904.

The work of Stassano and the successful production of ferro-alloys in the electric furnace were brought to the attention of the Canadian Government by Eugene Haanel, Government superintendent of mines. As is well known, Canada is not so favored with abundant high-grade ores and good coking coal as is the United States, and so it seemed to Haanel and his associates that the electric furnace might lead to the solution of their problem in Canada, namely, the treatment of low-grade ores, and of those ores high in phosphorus and sulphur.

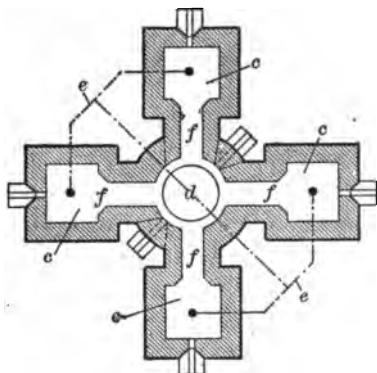
Ultimately, through the efforts of Haanel, a commission was appointed "to proceed to Europe for the purpose of investigating

^a Stansfield, A., *The electric furnace*, 1907, p. 13.

and reporting upon the different electrothermic processes employed in the smelting of iron ores and the making of the different classes of steel, then in operation or in process of development, in Italy, France, and Sweden."

This commission left Ottawa for England on January 21, 1904. When the commission arrived in England, F. W. Harbord, the well-known metallurgist, was engaged to act as metallurgist to it. Although the commission visited several plants in Europe, yet in none of them was direct smelting being attempted, nor were most of them in any way adapted for making experimental tests of direct smelting.

While the commission was at La Praz, France, Dr. Héroult was kind enough to demonstrate that it was possible to produce iron from the ore in an electric furnace.



EXPERIMENTS WITH KELLER FURNACE,
LIVET, FRANCE.

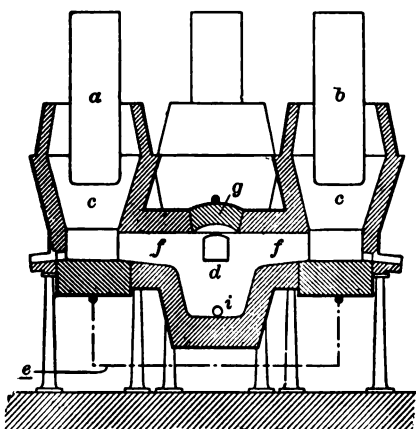


FIGURE 1.—Sectional plan and elevation of Keller furnace. Used by the Canadian commission in making tests at Livet, France.

At Livet, France, however, the commission found at the works of Keller, Leleux and Co. an electrical installation well suited for smelting tests of iron ores. Figure 1 shows the type of furnace used for making these tests. It consisted of two or more furnaces with vertical electrodes, connected by a central well, the current flowing from each furnace through the vertical electrodes *a* and *b*. In the tests

made by the commission the iron ore, fluxes, and coke were broken to such a size that all would pass through a $1\frac{1}{2}$ -inch ring. They were then mixed and "charged into the furnace in the annular space between the electrode and the walls of the furnace." In this connection it should be noted that the report of the commission clearly states that in these experiments the reduction was accomplished by means of solid carbon, as is shown by the following statement: "The heat, generated rapidly, raised the temperature and en-

abled the carbon mixed with the ore to reduce it to the metallic state, and as the temperature rose the metal fused and collected on the sole of the furnace." Later reference to this point is made. The gases resulting from the operation were allowed to escape and burn at the top of the furnace.

At Livet the commission made two separate experiments, the first lasting 55 hours and the second 48 hours. In conducting these experiments the materials were weighed and the weights checked. Especial attention was also given to such other points as were necessary to obtain accurate data in regard to the following matter:

The output of pig iron for a given consumption of electric energy.

The yield of metal per ton of ore charged.

The quantity of coke required as a reducing agent.

The quality of pig iron obtained, with especial reference to its suitability for (a) steel manufacture by (1) Bessemer or Siemen's acid process or (2) Bessemer or Siemen's basic process; and (b) foundry purposes.

The tests, performed on March 19, 20, and 21, 1904, marked the real beginning of the present practice of the reduction of iron ores in an electric furnace, for although, as will be seen later, the furnaces in use for this purpose at present differ considerably in construction from the Keller furnace, the principle employed is the same—namely, the ore, flux, and reducing agent are fed around the electrodes, and the heat generated by the current passing through the charge raises the temperature of the ore and fluxes to their melting point, and thus enables the carbon to reduce the ore with which it is mixed.

PROBLEMS NOT SOLVED BY COMMISSION OF 1904.

After the report* of the commission had been carefully studied it was evident that further data would have to be obtained in order to establish with some degree of exactitude the quantity of electric energy required per ton of product and the consumption of electrode. Also, the following important questions referring to Canadian conditions were either not taken up or were left in doubt by the Livet experiments:

Could magnetite, which is Canada's chief ore and is to some extent a conductor of electricity, be successfully and economically smelted by the electrothermic process?

Could iron ores with comparatively high sulphur content, but not containing manganese, be made into pig iron of marketable composition?

The experiments made at Livet with charcoal as a reducing agent in substitution for coke having failed, could the process be so modified that charcoal could be substituted for coke?

* Haanel, E., Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in operation in Europe: Mines Branch, Department of Interior, Canada, 1904.

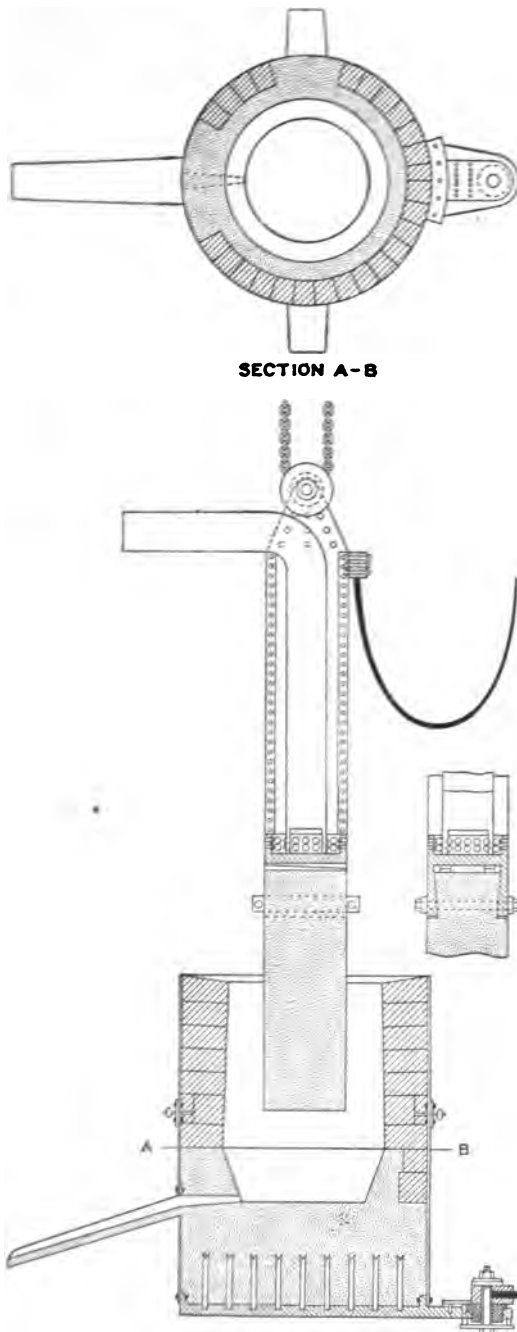


FIGURE 2.—Experimental furnace used at Sault Ste. Marie, Ontario.

The last problem was especially important, as charcoal and possibly peat coke would constitute Canada's home products, whereas coal coke for metallurgical processes has to be imported into several of the Provinces.

EXPERIMENTS AT SAULT STE. MARIE, ONTARIO, 1906.

A sum of \$15,000 was placed at the disposal of the Canadian commission for prosecuting experiments at Sault Ste. Marie, Ontario, in 1906. Owing to the advantages offered, the experimental furnace^a was erected at the plant of the Lake Superior Corporation at Sault Ste. Marie, Ontario, under the direction of Erik Nystrom, a member of the staff of the mines branch of the interior department, who was later prominently connected with the work at Trollhättan, Sweden. The furnace is shown in section in figure 2.

The experiments at Sault Ste. Marie were continued for several weeks. The report made to the Canadian Government by the superin-

^a Haanel, Eugene, Report on the experiments made at Sault Ste. Marie, Ontario, under Government auspices, on smelting of Canadian iron ores by the electrothermic process: Mines Branch, Department of Interior, Canada, 1907, p. 2.

tendent of mines contained substantially the following conclusions relative to the experiments:

Canadian magnetite ores can be as economically smelted as hematites by the electrothermic process.

Ores of high sulphur content can be made into pig iron containing only a few thousandths of 1 per cent of sulphur.

The silicon content can be varied as required for the class of pig that is to be produced.

Charcoal, which can be cheaply produced from mill refuse or wood that could not otherwise be utilized, and peat coke can be substituted for coke without being briquetted with the ore.

A ferronickel pig practically free from sulphur and of fine quality can be produced from roasted nickeliferous pyrrhotite.

Titaniferous iron ores containing up to 5 per cent of titanium can be successfully treated by the electrothermic process. This conclusion is based upon an experiment made with an ore containing 17.82 per cent of titanic acid, yielding a pig iron of good quality.

The electrical horsepower-year used per ton of pig produced during these experiments was about 0.277. As noted later, an average of 18 months' running at Trollhättan, Sweden, indicated that the electrical horsepower-year used per ton of pig produced was 0.340.

After the Government had discontinued its tests the experimental plant was acquired by the Lake Superior Corporation, and for a time was employed for the semicommercial production of ferro-nickel pig, but up to the present time no other electric pig-iron reduction furnace plants have been installed in Canada. This lack of plants is not because the reduction of iron ores in the electric furnace has not met the expectations of the commission, but because the peculiar economic and geographic conditions in Canada have not seemed to warrant, to the present time, the introduction of the electric furnace for the production of pig iron.

DEVELOPMENT OF THE ELECTRIC IRON-REDUCTION FURNACE IN SWEDEN.

The development of the electric iron-reduction furnace in Sweden has been due to the following reasons:

In Sweden the conditions are somewhat analogous to those found in Canada, that is, there are iron ores, but no coal for coking. There is, however, this difference between the conditions in Canada and in Sweden—in Sweden, as also in California, the ores are for the most part high in their iron content and rather free from impurities. Moreover, as has been indicated by Sundbarg^a, the iron industry has been well established in Sweden for some hundreds of years. The ores have been smelted in a blast furnace, charcoal for the most part having been used as a reducing agent, and Swedish charcoal iron is known the world over for its purity. However, it has long been

^a Sundbarg, A. G., Sweden, its people and its industries, 1904.

apparent to those familiar with the situation that there would have to be some innovation in order to enable Swedish iron manufacturers

to keep the cost of production down. Each year has seen an increase in the cost of charcoal, because wood for making charcoal is becoming scarce owing to the depletion of the forests and the increased production of wood pulp. These facts caused Swedish engineers to turn their attention to the possibilities of electric smelting.

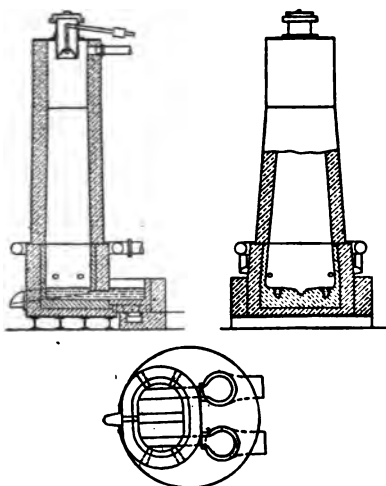


FIGURE 3.—First type of shaft furnace used in experiments of Grönwall, Lindblad, and Stalhane.

EXPERIMENTS OF GRÖNWALL, LINDBLAD, AND STALHANE, DOMNARFVET, SWEDEN.

were conducted by Grönwall, Lindblad, and Stalhane. The types of furnaces used by them are shown in figures 3, 4, and 5. The construction of the furnace shown in figure 3, is described by Yngstrom^a as follows:

It was a shaft furnace with the hearth lined with stamped silica. In the bottom of the hearth there were three channels: One in the middle leading to the tap hole for the pig iron, and one on each side. The latter communicated with two receptacles placed outside the shaft and filled with iron. The bottom of these receptacles was formed by blocks of granite packed on copper plates and connected with the electric-current supply. The furnace

The first electric-furnace smelting experiments made in Sweden

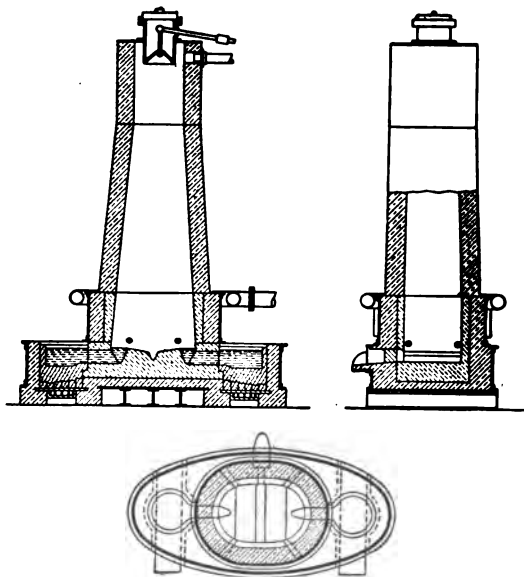
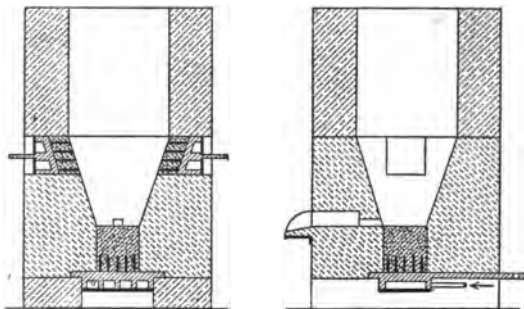


FIGURE 4.—Second type of shaft furnace used by Grönwall, Lindblad, and Stalhane.

was started with an air blast as an ordinary blast furnace, and when sufficient iron had collected on the hearth the air blast was cut off and the electric current turned on. The

^a Yngstrom, Lars, *Electric production of iron from iron ore at Domnarfvet, Sweden: Engineer (London)*, vol. 109, Feb. 25, 1910, p. 206.

idea was that the current entering the furnace through the iron by one of the channels would melt the charge in its passage to the opposite channel. It was possible to work the furnace in this manner during short periods, but it proved impossible to make the hearth durable. In part this was due to the fact that at the high temperatures used the silica became conductive of electricity. The furnace was therefore reconstructed in the manner shown in figure 4. The general principle of the design was the same as in the first furnace, but the lower portion of the furnace was of more solid construction. The electric current was introduced at opposite sides of the furnace. The most important alteration was, however, that the hearth was lined with magnesite brick. In consequence the hearth lasted somewhat better than before, but, in general, the same difficulties remained, and even the magnesite proved to be a fairly good conductor of electricity at high temperatures.



Finding that the types of furnaces above described were not suited to the work of reduction, Grönwall, Lindblad, and Stalhane turned their attention to the development of the type that later proved success-

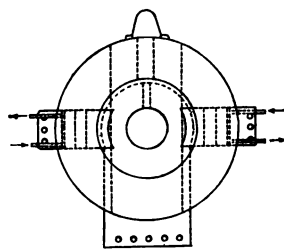


FIGURE 5.—Third type of furnace used by Grönwall, Lindblad, and Stalhane.

ful and is now being successfully operated in Sweden and Norway. In the work of developing this furnace to a feasible and commercial success these men were ably assisted by the engineers and capitalists of Sweden. An agreement was made with the Trafikaktiebolaget Grangesberg Oxelosund which enabled them to carry out their experiments on a large scale at the Domnarfvet Iron Works. Although the preliminary work on the furnace at Domnarfvet was begun in April, 1906, the furnace was not operated until April, 1907. Experimental work was conducted with it during the summer of 1907. The furnace is shown in figure 6. As the shaft of the furnace is low and open at the top, large quantities of charcoal were consumed at the top of the open shaft, and the gas escaping from the furnace consisted almost entirely of carbon monoxide. Based on the data

obtained during the operation of this furnace, a new furnace was constructed. This furnace, which is shown in figure 7, was higher, was closed at the top, and was so constructed as to permit the return to the crucible of a part of the gas

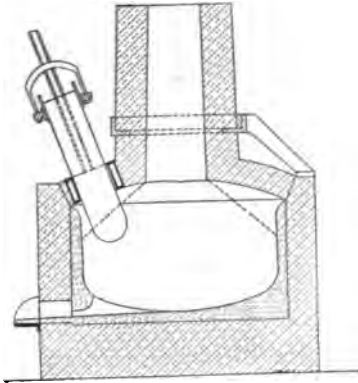


FIGURE 6.—First type of furnace used at Domnarfvet, Sweden.

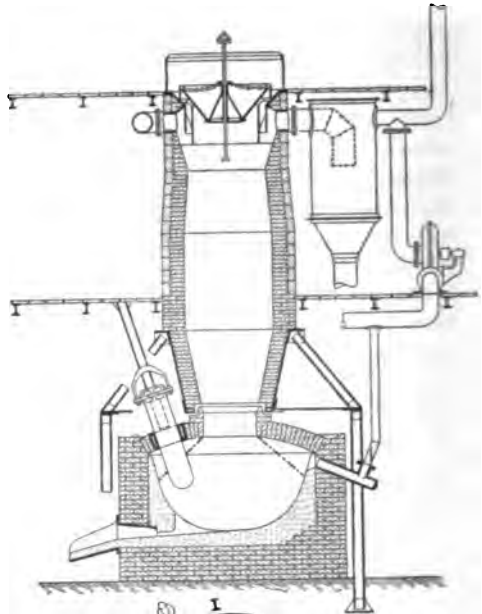


FIGURE 7.—Second type of furnace used at Domnarfvet, Sweden.

passing from the top of the shaft. The furnace has been described in detail by Yngstrom in his report.^a

EXPERIMENTS OF THE JERN-KONTORET, TROLLHÄTTAN, SWEDEN.

The experimental runs conducted at Domnarfvet satisfied those interested in the undertaking so well that it was decided to go a step farther and to construct and perfect a furnace of a size suitable for commercial purposes.

^a Yngstrom, Lars, Electric production of iron from iron ore at Domnarfvet, Sweden: *Engineer (London)*, vol. 109, Mar. 4, 1910, p. 237.

DESCRIPTION OF FURNACE.

This work was undertaken by the Jern-Kontoret, an association of the ironmasters of Sweden. Realizing the importance of the work to the iron industry of that country, the association voted \$90,000 for the purpose of putting up a plant and developing the process. The Swedish Government also assisted the project to the extent of furnishing power at a nominal figure from the plant at Trollhättan. Plans and sectional elevations of this furnace are shown in figures 8 to 11. The following description is taken from the official report delivered by the engineers to the Jern-Kontoret: "

The hearth is lined with "Hoganas" fire brick to a thickness of 360 mm. in the bosh and 450 mm. in the stack.

At the top of the shaft is a Tholander charging bell and cone, which is raised or lowered by means of a capstan driven by a motor.

The crucible rests on a concrete foundation. Like the shaft, it is surrounded by a sheet-iron shell $\frac{5}{8}$ of an inch thick. At the top this shell is reinforced by an iron band to take up the pressure of the arched roof.

The four electrodes project through the roof in a slanting position at an angle of 65° to the horizontal. At the openings in the roof the electrodes are surrounded by cooling jackets of copper provided at the top with asbestos packing

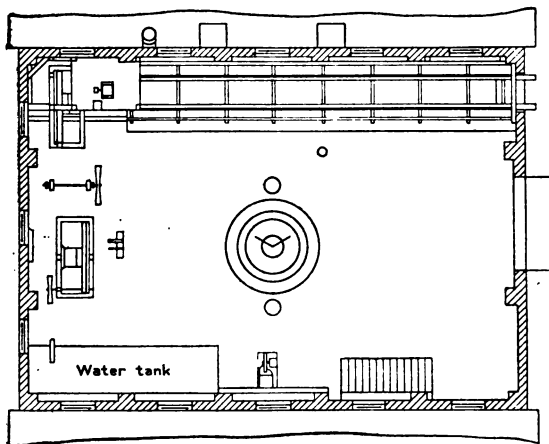
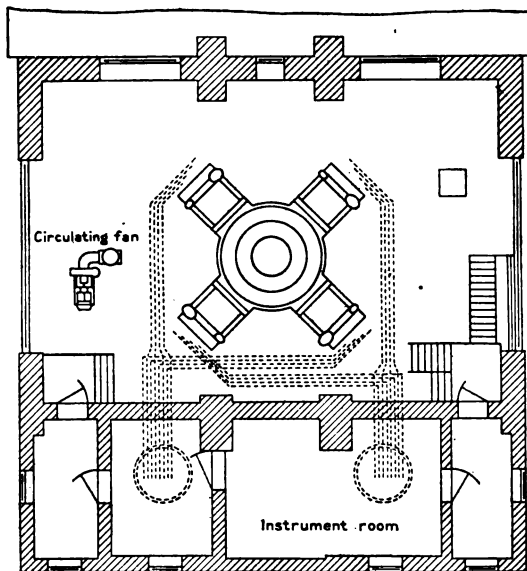


FIGURE 8.—Plan of furnace house at Trollhättan, Sweden.

* Leffer, J. A., and Odelberg, E., Redogörelse för Jern-Kontoret's Försöksverk i Trollhättan, May 31, 1911. Abstract: Iron and Coal Trades Rev., vol. 82, 1911, pp. 957, 1010.

to prevent the leakage of gas. The contacts for conducting the current to the electrode are arranged at the upper end and wedged between the electrodes and a holder of cast steel. This holder, in its turn, is supported on a frame movable up and down between two guides placed on each side of the electrode.

In this process provision is made for a circulation of the gases in such manner that gas is drawn by means of a fan from the gas outlets and blown into

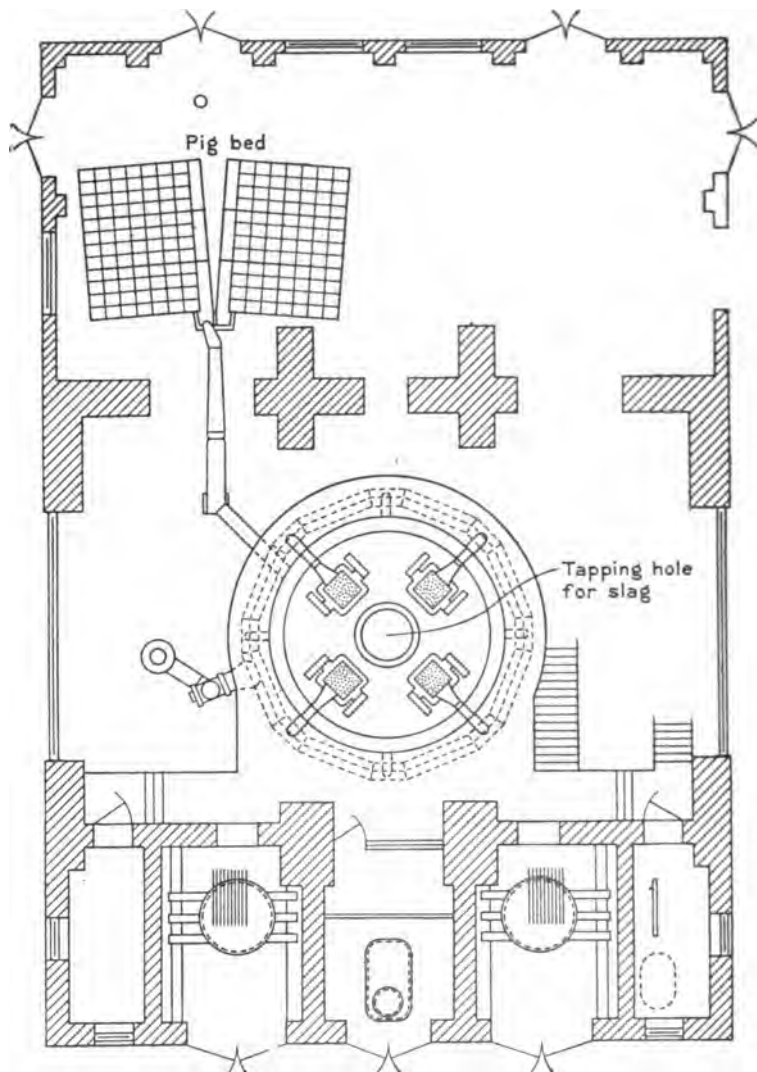


FIGURE 9.—Plan of furnace at Trollhättan.

the crucible. Consequently, the quantity of gas passing any given section of the shaft can be varied within certain limits by varying the speed of the fan.

This circulation of gas has two special objects, namely, (a) The gas blown into the crucible is there heated and gives off its heat to the charge in the shaft in passing up through it. Through this heating of the charge in the shaft a reduction of the ore by CO is facilitated, so that this gas is utilized to some

extent for the process. (b) The second purpose of the circulation of the gas is connected with the construction of the crucible, as the gas cools the roof and thus protects it from overheating. This cooling action of the gas is due in part to the absorption of heat or raising the temperature of the gas and in part to the decomposition of CO_2 and H_2O in the gas in contact with the incandescent carbon in the crucible.

A diagram of the electrical connections is shown in figure 12.

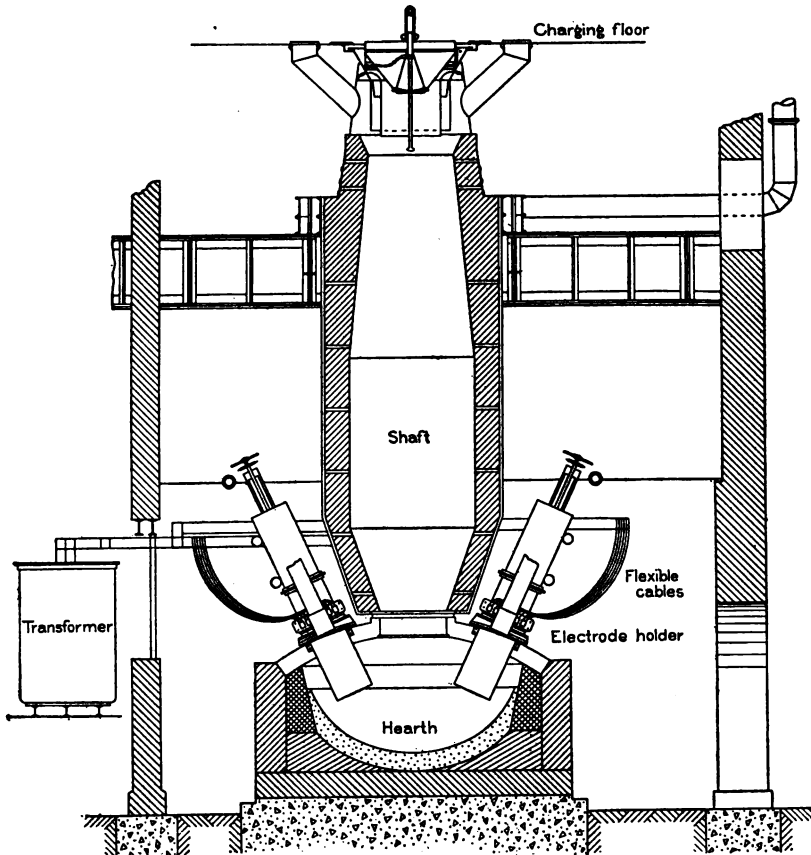


FIGURE 10.—Sectional elevation of furnace at Trollhättan.

CHANGES AFTER THE FIRST EXPERIMENTS.

Operations with this furnace were started November 15, 1910, and were continued without interruption until May 29, 1911. During this period various kinds of ore were smelted, and such grades of acid and basic slags were produced as seemed best suited to the treatment of each particular ore. The furnace worked well from the start, and was closed down that such alterations might be made as experience had demonstrated would be beneficial, and also that the crucible

might be relined and a new roof built, as the frequent alternation of acid and basic slags had damaged the brickwork of the crucible. The

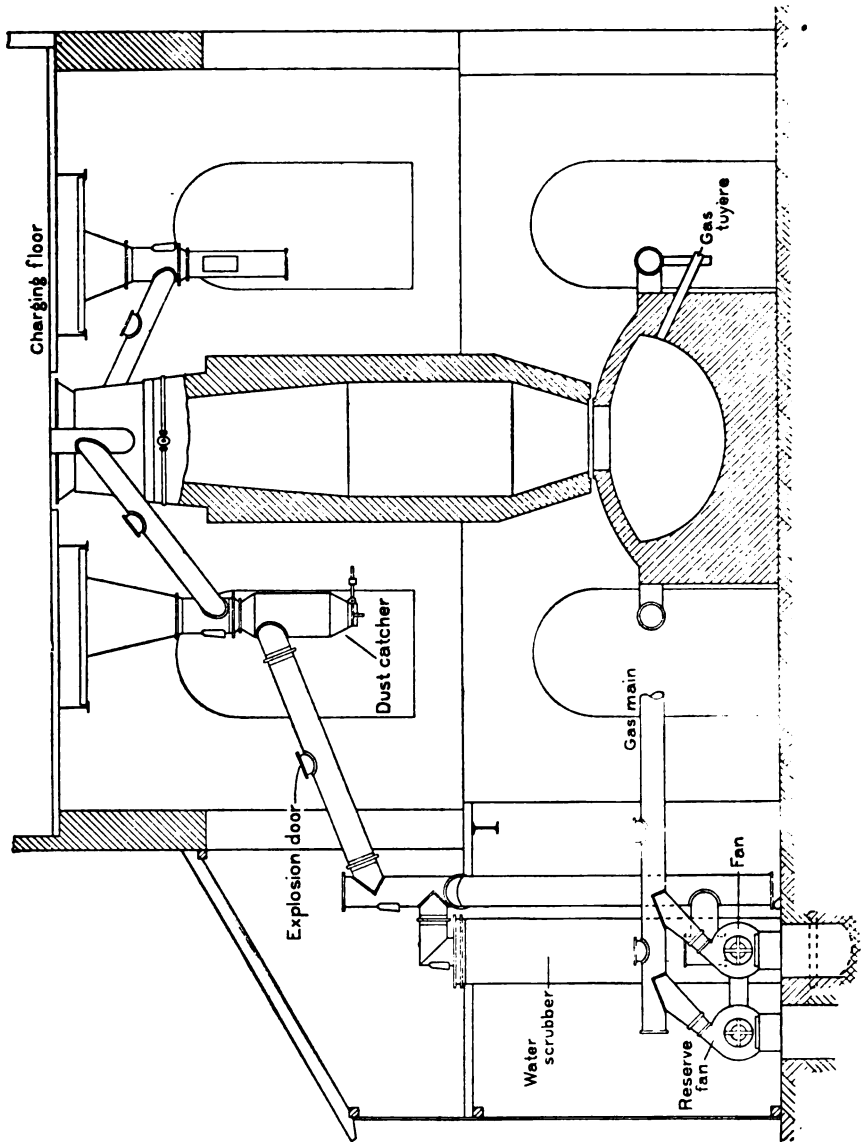


FIGURE 11.—Sectional elevation of furnace at Trollhättan, showing arrangement of gas circulation.

principal changes and additions that were made after the closing down of the furnace were as follows:

A system was installed for washing the gas which is returned to the crucible, so as to free the gas from dust particles before it reached the fan that caused the circulation.

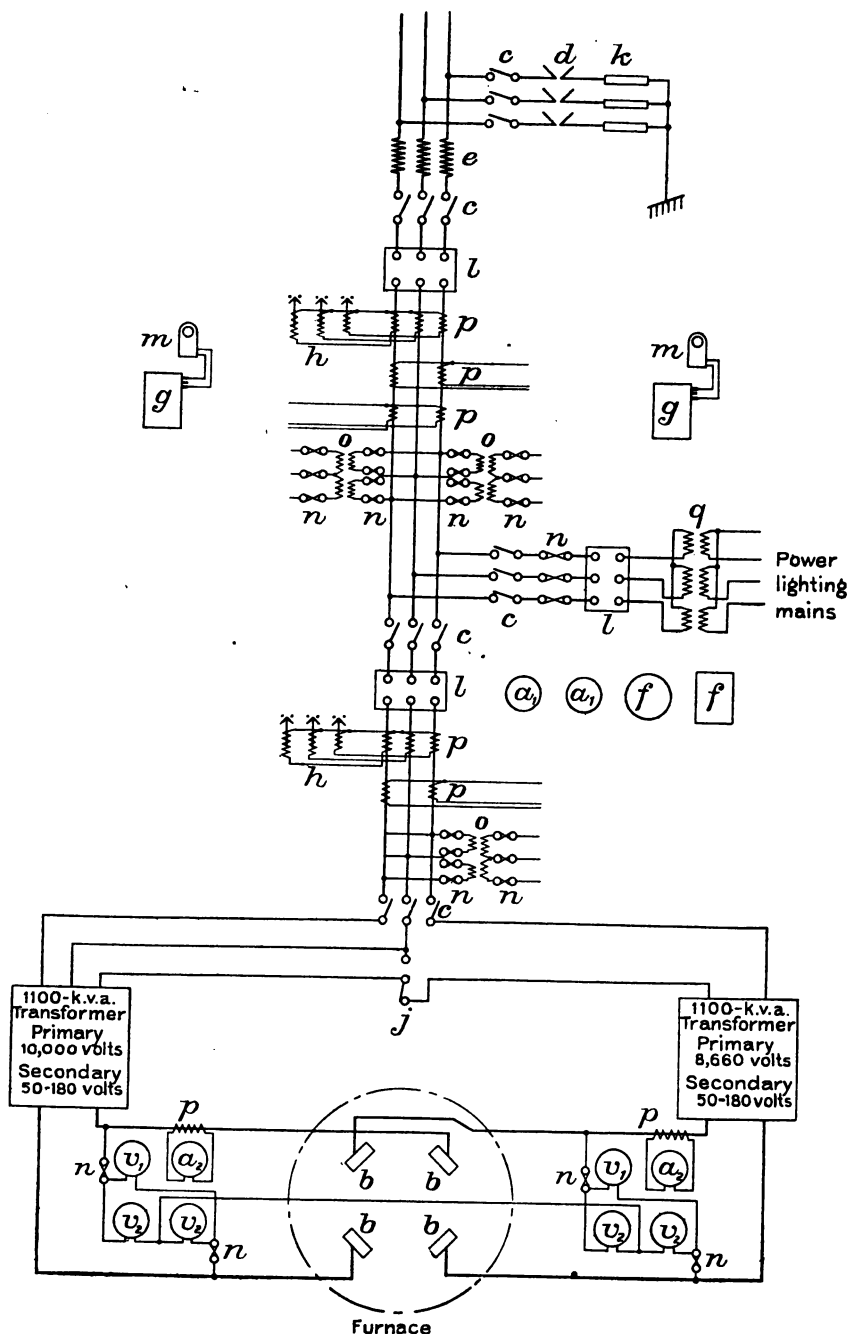


FIGURE 12.—Diagram of electrical connections of furnace at Trollhättan. *a*, ammeters; *b*, electrodes; *c*, main switches; *d*, lightning arresters; *e*, induction coil; *f*, kilowatt meters; *g*, kilowatt-hour meters; *h*, maximum relay; *j*, switch; *k*, oil resistance; *l*, automatic switches; *m*, clock control; *n*, fuses; *p*, *q*, transformers; *v*, voltmeters.

Electrode contact jackets were constructed and installed which permitted the electric current to be introduced to the electrodes at the point where they pass through the roof of the crucible rather than at the end of the electrode as was formerly done (fig. 10).

Round electrodes, 23½ inches in diameter, were substituted for square electrodes. The latter were built of four carbons, of about 12 by 12 inch section.

The electrodes were threaded so as to permit their being joined by means of screw nipples, thus avoiding the waste of butt ends, as formerly, as soon as the electrodes became too short for further use.

SUMMARY OF WORKING RESULTS AND ANALYSES.

The following tables present in condensed form the results obtained by the operation of the furnace from November 15, 1910, to April 9, 1911:

TABLE 1.—*Results of electric iron smelting at Trollhättan, Sweden.*

Item.	Nov. 15, 1910. ^a	Nov. 16, 1910, to Feb. 11, 1911.	Feb. 11 to Feb. 19, 1911.	Feb. 19 to Mar. 19, 1911.	Mar. 19 to Apr. 9, 1911.	Total.
Iron in ore, per cent.....	64.92	65.57	65.06	49.50	57.92	61.54
Iron in charge, per cent.....	59.80	62.10	62.56	42.42	53.06	57.00
Slag per ton of iron, kg.....	390.00	205.00	224.00	780.00	458.00	327.00
Material charged per hectoliter of charcoal, kg.....		66.49	71.13	90.31	69.88	70.77
Charcoal per ton of iron, hectoliters.....		24.22	22.47	26.10	26.97	24.79
Charcoal contents:						
Water, kg.....		69.1	50.8	59.8	40.2	60.9
Gas, kg.....		41.7	36.9	49.3	43.1	42.9
Ash, kg.....		11.8	11.0	13.2	17.2	12.8
Coke, kg.....		293.1	277.6	323.4	325.7	301.4
Total, kg.....		415.7	376.3	445.7	426.2	418.0
Time consumed in working, hours, minutes.....	7 50	2,009 56	184 32	639 18	506 34	3,348 10
Time consumed in interruptions, hours, minutes.....		105 39	4 58	20 57	22 11	153 45
Total time, hours, minutes.....	7 50	2,115 35	189 30	660 15	528 45	3,501 55
Average load, kilowatts.....	1,121	1,319	1,694	1,017	1,733	1,344
Total kilowatt-hours used.....	8,780	2,661,029	312,601	650,480	877,706	4,500,596
Kilowatt-hours per ton of iron.....	3,800	2,296	2,149	2,623	2,643	2,391
Iron per kilowatt-year, tons.....	2.31	3.82	4.08	3.34	3.31	3.66
Gross electrode consumption, kg.....		13,012	1,578	2,281	2,474	19,345
Net electrode consumption, kg.....		6,743	763	1,121	1,285	9,912
Gross electrode consumption per ton of iron, kg.....		11.24	10.84	9.19	7.45	10.28
Net electrode consumption per ton of iron, kg.....		5.83	5.24	4.52	3.87	5.27

^a Furnace being filled.

^b Term "net" represents 51.24 per cent of gross.

TABLE 2.—Analyses ^a of ore and limestone used in electric iron smelting plant at Trollhättan, Sweden.

[From Iron and Coal Trades Review, June 9, 1911.]

Number of sam- ple. ^a	Oxides.										Loss on ignition.	Elements ^b					
	Fe ₂ O ₃	FeO	MnO	MgO	CaO	Al ₂ O ₃	TiO ₂	SiO ₂	P ₂ O ₅	S		Total.	Fe	Si	Mn	S	P
.....	P. ct. 0.24	P. ct. 91.05	P. ct. 0.35	P. ct. 0.31	P. ct. 54.32	P. ct. Trace.	P. ct. 0.10	P. ct. 1.68	P. ct. Trace.	P. ct. 0.001	P. ct. 42.94	P. ct. 97.811	P. ct. 0.17	P. ct. 0.79	P. ct. 0.27	P. ct. 0.001	P. ct. Trace
.....	1.41	64.80	7.30	1.12	3.28	0.16		3.42	0.050	0.011		99.921	66.04	1.61	0.12	0.001	0.022
.....		25.07	4.0	3.17	3.95	1.16		8.45	0.016	0.019		99.685	49.18	3.97	5.66	0.019	0.07
.....	53.36		18	60	71	59		20.13	0.034	0.013		101.177	55.45	9.47	31	0.013	0.015
.....	89.85	2.49	14	85	99	42		5.08	0.014	0.007		100.471	64.80	2.39	14	0.007	0.006
.....	5.52		1.84	2.57	53	06	48	3.47	0.037	0.012		99.679	67.10	1.63	11	0.012	0.016
.....	80.88	37.45	63	1.57	55	1.82		7.31	0.032	0.017		99.459	59.68	3.44	49	0.017	0.014
.....		79.35	4.5	4.23	2.31	2.55		13.25	0.028	0.023		99.501	58.51	3.68	35	0.023	0.012
.....		73.10	22	4.73	4.29	2.67		11.19	0.025	0.014		99.539	53.05	6.23	17	0.014	0.011
0.....		73.10	19	6.90	4.37	1.87		16.28	0.025	0.009		99.304	52.90	5.26	15	0.011	0.011
1.....		71.69	12	7.35	1.34	1.37		15.05	0.027	0.009		99.356	51.88	7.08	11	0.009	0.011
2.....		78.02	14	13	5.82	1.34		10.63	0.029	0.011		101.830	50.71	5.00	10	0.013	0.013
3.....	2.92	70.13	12	7.90	7.05	1.90		15.03	0.009	0.004		101.103	49.34	10.33	09	0.004	0.004
4.....		68.18	12	6.17	2.88	1.77		21.97	0.009	0.002		101.011	50.41	12.98	13	0.002	0.004
5.....	62.22		17	70	44	1.06		11.14	0.025	0.011		99.766	51.42	5.23	32	0.011	0.011
6.....		71.11	41	3.60	7.98	2.65		8.28	0.009	0.020		99.539	56.76	3.19	29	0.020	0.012
7.....		78.50	37	4.68	3.34	2.26		21.96	0.027	0.002		100.869	52.69	10.31	26	0.002	0.004
8.....	48.24		98	82	1.29	1.55		12.98	0.009	0.021		99.370	60.85	6.09	26	0.021	0.004
9.....		72.57	34	4.90	4.38	1.96		7.96	0.009	0.004		99.333	51.88	3.74	13	0.004	0.006
10.....	83.53	58	17	2.53	2.81	1.49		13.35	0.014	0.052		100.434	46.31	6.27	27	0.052	0.006
11.....	75.60		35	5.32	4.68	2.12		4.09	0.021	0.013		100.434	46.31	1.92	12	0.013	0.008
12.....	90.42	1.07	15	2.54	78	1.02	20	15.15	0.018	0.011		99.109	48.37	7.11	29	0.011	0.005
13.....	65.14	1.63	37	7.02	5.19	1.56		15.98	0.011	0.023		99.094	49.75	7.50	22	0.023	0.005
14.....	67.59		28	4.75	6.10	1.96		21.25	0.066	0.051		101.077	45.06	9.97	28	0.051	0.009
15.....	62.11	1.12	36	7.08	6.25	2.33		12.05	0.021	0.055		99.916	53.25	5.66	34	0.055	0.009
16.....	72.44	1.12	44	6.20	4.03	1.40		3.35	0.025	0.001		100.376	66.90	1.57	10	0.001	0.011
17.....	91.98		13	2.33	59	1.58	79										

^a Analyses made at the Degerfors Iron Works laboratory.^b Samples taken at the works.^c CO₂, 7.02; C, 0.86.

TABLE 3.—Analyses of iron, slag, and furnace gas.

INCOMPLETE ANALYSIS OF IRON.

Date.	Number of charge.	C	Si	Mn	S	P
1911.		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Jan. 3.....	138	4.19	1.35	0.90	0.004	0.021
Jan. 14.....	171	4.04	.75	.82	.005	.020
Mar. 16.....	324	3.00	.14	.08	.023	.019
Mar. 30.....	372	3.10	.45	.40	.014	.010

ANALYSIS OF SLAG.

Date.	Number of charge.	SiO ₂	Al ₂ O ₃	TiO ₂	FeO	MnO	CaO	MgO	CaS	P ₂ O ₅	Total
1911.		<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>
Jan. 3.....	138	44.76	1.20	2.58	1.75	1.35	31.70	15.10	0.077	Trace.	98.517
Jan. 14.....	171	41.60	6.85	2.72	1.49	1.48	28.91	16.70	.063	0.000	99.313
Mar. 16.....	324	46.82	5.06		6.89	.23	33.27	7.97	.023	.041	100.274
Mar. 30.....	372	37.98	6.98	.37	1.28	.52	27.98	23.45	.123	.000	98.683

INCOMPLETE ANALYSIS OF SLAG.

Date.	Number of charge.	Fe	Si	Mn	S	P
1911.		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Jan. 3.....	138	1.36	21.05	1.05	0.034	Trace.
Jan. 14.....	171	1.16	19.56	1.14	.028	0.000
Mar. 16.....	324	5.36	21.97	.18	.009	.018
Mar. 30.....	372	1.00	17.82	.40	.064	.000

ANALYSIS OF FURNACE GAS, BY VOLUME.

Date.	Number of charge.	CO ₂	O	CO	H	CH ₄	N
1911.		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Jan. 3.....	138	28.2					
Jan. 14.....	171	27.2	0.0	57.5	14.8	0.0	0.5
Mar. 16.....	324	12.6		71.9	13.0	1.7	.8
Mar. 30.....	372	19.2		59.7	17.6	2.5	1.0

REPORT ON OPERATION OF THE FURNACE FROM AUGUST, 1911, TO MAY, 1912.

The furnace was again put into operation in August, 1911, and was in continuous operation up to July, 1912, when it was relined. In May, 1912, a report was submitted to the Jern Kontoret by the engineers in charge of the operation of the furnace from August, 1911, up to that time. The following summary is taken from an abstract of the report that appeared in Metallurgical and Chemical Engineering, July, 1912, p. 413:

Results of operation of furnace from August, 1911, to May, 1912.

Iron, in ore, per cent.....	60.95
Iron, in burden, per cent.....	66.84
Weight of slag per ton of iron, kg.....	323

Weight of fuel per ton of iron, kg-----	404
Volts on furnace-----	73.6
Amperes on furnace-----	11,423
Power on furnace, kw-----	1,482
Power per ton of iron, kilowatt-hours-----	2,225
Output of iron per kilowatt-year, tons-----	3.94
CO ₂ in throat gas, per cent-----	23.49
Volume of circulating gas, cubic meters per second-----	0.24
Pressure of gas in furnace, mm. water-----	225
Temperature at bottom of shaft, °C-----	441
Temperature at middle of shaft, °C-----	279
Temperature at top of shaft, °C-----	17
Electrode consumption, per ton of iron:	
Gross, kg-----	5.72
Net, kg-----	5.18

The variation in the charcoal and power consumption per ton of iron produced, depending upon the iron content of the ore used, is shown in the following tabulation, taken from the article mentioned above:

Variation in power and charcoal consumption.

Fe in ore, per cent-----	67.66	66.71	57.15	54.83
Slag per ton of iron, kg-----	166	209	409	521
Coal per ton of iron, kg-----	347	380	439	490
Kilowatt-hours per ton of iron-----	1,819	2,253	2,369	2,525

The composition of the pig iron produced varied between the following limits:

Variation in composition of pig iron.

Element.	Range.		Average.
	Low.	High.	
Carbon-----	<i>Per cent.</i> 2.688	<i>Per cent.</i> 3.859	<i>Per cent.</i> 3.406
Silicon-----	.183	2.473	.725
Manganese-----	.190	1.366	.477
Sulphur-----	.0037	.0341	.0126
Phosphorus-----	.0134	.0454	.0200

The composition of the gases produced varied as follows:

Variation in composition of gases.

Gas.	Range.		Average.
	Low.	High.	
CO ₂ -----	<i>Per cent.</i> 14.63	<i>Per cent.</i> 31.80	<i>Per cent.</i> 23.49
CO-----	31.80	71.10	63.15
H ₂ -----	7.87	14.50	10.35
CH ₄ -----	.82	2.37	1.52
N ₂ (difference)-----	.50	3.41	1.49

The ratio of CO to CO₂ by volume varied from 4.8 to 1.8. The calorific power of the gas per cubic meter varied from 2,035 to 2,569, an average of 2,297 calories, or from 57.5 to 72.7 calories per cubic foot.

The oxygen ratio of the slags produced varied from 2.09 to 1.23. Analysis of the slags showed the following variations in composition:

Variation in composition of slags.

	Per cent.
SiO ₂	36.54 to 45.15
Al ₂ O ₃	1.96 to 9.20
TiO ₂	Trace to 7.42
FeO.....	.58 to 6.22
MnO.....	.19 to 6.00
CaO.....	21.08 to 45.58
MgO.....	6.22 to 26.65
CaS.....	.04 to .44
P ₂ O ₅	Trace to .034

The temperatures of the iron and the slag issuing from the furnace varied as follows:

Iron, °C.....	1,230 to 1,420
Slag, °C.....	1,200 to 1,460

In September, 1912, the Jern-Kontoret leased the plant at Trollhättan to the Strömsnäs Iron Works, which has since worked it as a commercial plant. The final report on the research work at Trollhättan, as well as a report on similar commercial installations elsewhere, has now been published.^a A summary of the report, as furnished by Electro-Metals, of London, is largely presented below.

Use of concentrates.—The proportion of concentrates ought not to exceed 20 per cent of the ore charged. This figure, however, does not appear to be final, as in a somewhat modified furnace of the same type constructed later at the plant of the Uddeholm Co. at Hagfors 25 per cent of concentrates is used without any difficulty.

Power consumption.—The power consumption per ton of pig iron varies in proportion to the iron content in the ore. A poor ore and a pig iron high in silicon and manganese require more power than rich ore and pig iron low in silicon and manganese. For such iron the power consumption averages only 2,067 kilowatt-hours per ton of pig iron—that is, there is obtained 4.22 tons of pig iron per kilowatt-year, or 3.10 tons per horsepower-year.

Charcoal consumption.—The charcoal consumption per ton of pig iron varies from 20 to 24 hectoliters (57 to 68 bushels), depending on the quality of the charcoal and the charge. Coke, unless mixed with charcoal, is unsuitable for this furnace.

^aAbstract of report, Iron and Coal Trades Rev., vol. 86, 1913, p. 714.

Consumption of electrodes.—The consumption of electrodes at Trollhättan has been reduced to less than 3 kilograms per ton of pig iron. At Hagfors it has amounted to as much as 6 to 9 kilograms. This discrepancy is explained by the fact that the electrode consumption is increased in proportion to the higher power consumption for a poor charge, and is further increased by the more efficient circulation of gases and higher carbon dioxide content in the gas. The lower electric load per unit of surface at the Hagfors furnaces also contributes to the higher electrode consumption at this plant.

Cost of repairs.—The cost of repairs is lower than was at first expected. In the manufacture of pig iron containing silicon and manganese the cost for repairs is higher than in producing pig iron with a low content of those elements.

Quality of the pig iron.—The silicon content does not vary more than in the product of an ordinary blast furnace. The phosphorus content is lower than when the same quality of charge is used in an ordinary blast furnace, owing to the lower consumption of charcoal in the electric furnace. The sulphur content is, however, slightly higher in the product of the electric furnace. However, it should be observed that both at Trollhättan and at Hagfors unroasted ores have been used without any difficulty arising from the sulphur present.

The quality of the pig iron from the electric furnace has been highly commended. It acts particularly well in the open-hearth furnace, and steel made from it is certainly not inferior to steel made from ordinary pig iron. E. Odelberg, managing director of the Strömsnäs Iron Works, states that the electric pig iron is "of the very best quality for the open-hearth process and, as regards the uniformity of the silicon content, fully as uniform as iron from an ordinary blast furnace." A. Herleinius, managing director of the Uddeholm Co., states that "the electric pig iron has been used with satisfactory results, both for the open-hearth, Bessemer, and Lancashire processes. Generally speaking, there has been no difficulty in obtaining pig iron of uniform quality, although slightly better uniformity may possibly be obtained with a very carefully conducted blast furnace."

Value of the gas.—At the plant of the Uddeholm Co., at Hagfors, the gas from the furnaces has been used with very good results for heating the open-hearth furnaces. It is estimated that the value of the gas obtained per ton of pig iron may be taken at 2.50 kroner (about \$0.67).

With regard to the furnace that was used, the report expresses the opinion that its construction and general dimensions have been found to be suitable. A summary of the most important figures relating to the economic results is presented in Table 4 following.

TABLE 4.—Results of electric iron smelting with furnace at Trollhättan.

Item.	Nov. 15, 1911, to May 29, 1911.	Aug. 4, 1911, to June 21, 1912.	Aug. 12, 1912, to Sept. 30, 1912.	October to December, 1912.
Ore, concentrates and briquets, kg.....	4,336,338	7,917,214	a 1,406,530	2,914,830
Limestone, kg.....	345,406	647,479	108,150	169,944
Charcoal, hectoliters.....	65,474.5	107,282.5	21,859.5	44,934.5
Coke, kg.....		70,854		
Electric energy, kw.-hours.....	6,339,131	10,845,180	1,939,073	3,957,565
Iron content of ore, per cent.....	60.79	60.75	68.67	65.38
Iron produced, kg.....	2,636,098	4,809,670	965,915	1,905,865
Slag per ton of iron, kg.....	350	324	192	
Electrodes, per ton of iron, gross kg.....	10.00	6.08	3.02	2.78
Electrodes per ton of iron, net kg.....	4.95	5.17	3.02	2.78
Charcoal per ton of iron, hectoliters.....	24.84	22.31	22.63	23.58
Working time, hours, minutes.....	4,441 20	7,218 23	1,173 8	2,158 30
Repairs, hours, minutes.....	236 53	506 07	13 47	49 30
Repairs, percentage of total time.....	5.06	6.55	1.16	2.24
Average load, kw.....	1,427	1,502	1,653	1,833
Kilowatt-hours, per ton of iron.....	2,406	2,255	2,007	2,076
Iron per kw.-year, tons.....	3.64	3.88	4.36	4.22
Iron per hp.-year, tons.....	2.68	2.86	3.20	3.10

a 1,209,825 kg. represented Kåfuna "A" ore containing 69.61 per cent of Fe.

The table shows that step by step the results have been improved, the quantity of iron per horsepower-year increased, whereas the electrode consumption and time for repairs have been reduced, these three items in conjunction with the saving in charcoal being the decisive factors as regards electric iron smelting.

The figures show that during the last few months of the experimental work as much as 3.20 to 3.10 tons of iron were obtained per horsepower-year, whereas before the alterations were made the highest average figure was 2.86 tons. The highest expected yield was 3 tons per horsepower-year, which was therefore exceeded. It may also be mentioned that during single periods of several weeks when especially suitable ores were used, the highest average figures above mentioned were materially exceeded. It will thus be seen that as regards efficiency the results of the furnace surpassed expectations. The electrode consumption shows a remarkable decrease. The consumption of 13.8 kilograms during the first period of working was finally reduced to about 3 kilograms per ton of pig iron. The cost and time required for repairs were items that could not be estimated beforehand. Experience has shown that both the cost and the time required are less than could have been expected. Utilization of the gas, when practicable, should also be taken into account. At Hagfors the use of gas for firing the open-hearth furnaces is estimated to reduce the cost of the pig iron about 65 cents per ton.

DURABILITY OF THE ROOF OF THE CRUCIBLE.

In perfecting the electric furnace one of the most serious difficulties that had to be overcome was maintaining the brickwork of the crucible. In the operation of the furnace at Trollhättan from August 4, 1911, to June 21, 1912, the following repairs were made: October 26, 1911, the arch over the crucible was partly repaired;

February 13, 1912, the arch was entirely rebuilt; April 27, 1912, the arch was partly repaired. At Hagfors the arch was rebuilt after working four and one-half months.

COMMERCIAL FURNACES OF THE SWEDISH TYPE.

As a result of the successful working of the furnace at Trollhättan the following furnaces have been built:

Furnaces built subsequent to the Trollhättan furnace.

Place.	Number. of furnaces.	Horsepower of each.	Total horsepower.
Sweden:			
Domnarvret.....	1	3,500	3,500
Hagfors.....	2	3,000	6,000
Norway:			
Hardanger.....	2	3,500	7,000
Arendal.....	3	3,000	9,000
Switzerland.....	1	2,500	2,500

However, the furnaces at Hardanger, after having been operated for about nine months, were closed down, the reason given being that suitable raw materials were not available in Norway. In Sweden charcoal is used as a reducing agent, whereas in Norway only coke is available for that purpose. During the experimental work at Trollhättan, coke was tried as a reducing agent, but was found unsatisfactory, and the failure at Hardanger, where coke only was used, confirms the conclusions reached by the engineers at Trollhättan regarding the unsatisfactory results with the use of coke as a reducing agent.

The bus-bar connections and the arrangements of electrodes around the crucible in the Swedish type of electric shaft furnace are shown in figure 13.

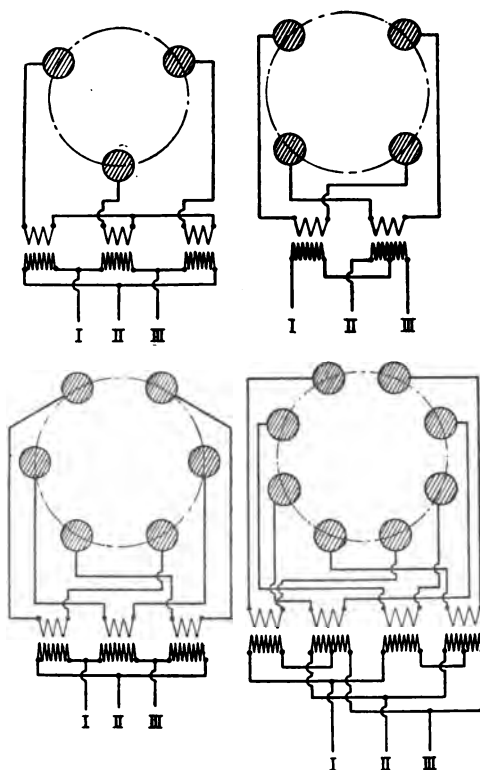


FIGURE 13.—Bus-bar connections and arrangement of electrodes in Swedish type of electric shaft furnace.

The Uddeholm Co., at Hagfors, Sweden, is adding a third furnace to its electric furnace installation at Hagfors, and is constructing

three similar furnaces at its Nykroppa works. It will probably substitute electric smelting altogether for its blast furnaces.

The Stora Kopparbergs Bergslags is also preparing to introduce electric smelting on a large scale, and according to reports, other iron works in Norway and Sweden will follow its example.

The cost of construction of the plant at Trollhättan is given below :

Cost of construction of electric iron smelting plant at Trollhättan, Sweden.

Preparation of site :

Excavations, foundations, and fencing; narrow-gage tram lines, 3 turntables, and 5 trucks; 1 5-ton and 1 40-ton weigh bridge; railroad siding (about 1,830 feet) with 1 turntable; earthenware water main from the canal (400 feet)-----	\$10,727.42
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Buildings :

Furnace house with 2 windlasses and inclined tracks	\$14,735.00
Charcoal storage house with conveyors and elevators	6,032.51
Crusher house and elevator-----	1,282.97
Office, laboratory, and storeroom-----	1,393.87
Repair shop with equipment-----	1,040.63
General storehouse-----	243.64
	24,728.62

Furnaces :

1 furnace for 2,500 horsepower (ironwork, castings, masonry, etc.)-----	13,117.12
Electric equipment-----	13,781.70
Bars and cables to the electrodes-----	3,381.93
Exhaust fan for furnace gases-----	826.94
Pump and motor (for water)-----	1,452.64
Water tank and pipes-----	942.19
	33,502.52

Crusher-----	1,010.80
Electric lighting and motors: Transformers and wiring; 4 motors for exhaust fan, crusher, ore and coal elevators, and conveyors: direct-current transformers-----	5,157.24
Laboratory equipment-----	889.88
Instruments (self-recording) for measuring temperature, pressure, and volume of gas-----	1,894.25
Tools and sundry supplies-----	1,212.30
Furniture and fittings for office, etc-----	675.00
Supervision and sundry expenses-----	5,637.86
Drawings, specifications, and license fee-----	7,500.00
Total-----	92,935.89

CHEMISTRY OF THE REDUCTION OF IRON ORES IN THE ELECTRIC FURNACE.

As is well known, the reduction of iron ores is effected by heating the ore and the reducing agent to such a temperature that a reaction takes place between the oxygen and the reducing agent, which usually is some form of carbon. For the most part, iron ores at the present time are reduced in the blast furnace. In order to get a clear idea of the difference between a blast furnace and an electric-reduction

furnace, let us first briefly note how the work of reduction and subsequent melting of the reduced iron and fluxes is brought about in the blast furnace.

The charge is composed of ore, oxide of iron plus gangue materials, fluxes for combining with the gangue materials of the ore in such proportions determined by analysis and calculation as will make a fusible slag, and the fuel, which is needed to provide heat and also acts as a reducing agent. This fuel is generally coke or charcoal and contains practically no volatile matter. In the tuyère zone the larger part of the fixed carbon passes to CO, but whether it first passes through the CO₂ state and is then reduced to CO by incandescent carbon is still a debated subject.^a At any rate CO is the final product, and in its production a high enough temperature is produced to melt down the previously reduced iron and to form a fluid slag from the fluxes and gangue materials present. At this point the difference between the blast furnace and an electric-reduction furnace should be noted, for in the electric furnace the heat necessary for melting the charge is furnished by the electric current. In the blast furnace the quantity of fuel necessary to produce the smelting temperature, and not the amount necessary for the reduction of the oxides, determines the amount of fuel that is used. In the electric furnace the amount of electric energy necessary for producing the smelting temperature is used in place of coke or charcoal; and as only one-third or less of the coke that is used in blast-furnace work is needed in the electric furnace for the reduction of the iron oxides, it is possible to produce in the electric furnace three times as much iron with 1 ton of coke or charcoal as in a blast furnace. Such being the case, the amount of electrical energy and carbon that are necessary to produce 1 ton of pig iron in an electric furnace may now be considered.

This problem has been carefully worked out by Yngstrom,^b Richards,^c and others, and hence it is necessary in this connection to give only the essential details in such a calculation.

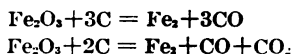
In determining the amount of electrical energy necessary to produce 1 ton of pig iron from a given ore the amount of heat absorbed by the following processes must be calculated: The reduction of the iron oxides to iron; the reduction of the SiO₂ to Si; the melting and superheating of n kilograms of iron; the melting and superheating of n kilograms of slag; the heating of x kilograms of CO₂ plus CO to whatever temperature it is determined that they should escape from the shaft.

^a Belden, A. W., Foundry-cupola gases and temperatures: Bull. 54, Bureau of Mines, 1913, pp. 14-19.

^b Yngstrom, Lars, Electric production of iron from iron ore at Domnarfvet, Sweden: Engineer (London), Feb. 25, 1910, p. 206.

^c Richards, J. W., Metallurgical calculations, pt. 2, 1907, p. 403.

From the number of calories absorbed in this manner may be deducted the heat that would be developed by the combustion of the x kilograms of carbon needed for the reduction. The calculation of the amount of carbon needed for the reduction of the oxide is not an easy matter, for, as has been pointed out by Richards, in calculating the amount of carbon necessary to reduce the oxide of iron there is no way of knowing what proportion of carbon will form CO and what proportion CO₂. The amount of each that will be formed depends upon the temperature, CO being almost the only product that is formed at a high temperature, whereas CO₂ is formed in increasing quantities as the temperature decreases, the reactions being represented as follows:



From these equations we note that if the gaseous product of the reduction is all CO, not more than one-third as much carbon is required for the reduction as is required in the case of blast-furnace practice. Yngstrom in his calculations assumes that when carbon combines with oxygen in the electric furnace a mixture is formed that contains 30 per cent (by volume) of CO₂, and on this basis he calculates the carbon necessary for the reduction of the iron oxides present and the subsequent development of heat by the formation of CO and CO₂ by the following equation:



In his computations he uses the following theoretical values:

- To reduce 1 kg. of Fe from Fe₂O₃ there is required 1,650 calories.
- To reduce 1 kg. of Fe from Fe₃O₄ there is required 1,800 calories.
- To reduce 1 kg. of Si from SiO₂ there is required 7,830 calories.
- By oxidation of 1 kg. of C to CO₂ there is developed 8,080 calories.
- By oxidation of 1 kg. of C to CO there is developed 2,470 calories.
- 1 kilowatt-hour corresponds to 857 calories.
- 1 kg. of pig iron requires for melting and superheating 280 calories.
- 1 kg. of slag (monosilicates) requires 595 calories.

As a result of his calculations it is seen that for the production of 1 ton of pig iron containing 3 per cent of carbon, 1 per cent of silicon, 96 per cent of iron, and traces of manganese, phosphorus, and sulphur, made from a charge containing 60 per cent iron in the form of Fe₃O₄ and with the escaping gases containing 30 per cent of CO₂, 248 kg. of carbon, or 292 kg. of coke, and 1,460 kilowatt-hours would be required, which would correspond to 4.4 tons of pig iron per horsepower year of 365 days.

The production of this amount of iron per kilowatt-year has not been attained in actual practice, for, according to the latest reports from Trollhättan, only 3.94 tons is produced per kilowatt-year. At this point it may be interesting to compare the number of heat

calories computed by Yngstrom as being necessary to produce 1 ton of pig iron with the number that can be calculated from Junge-Hermsdorf's "Heat Balance of a Blast Furnace of 250-ton Capacity." A furnace producing 250 tons in 24 hours will produce, in round numbers, 10 tons an hour, and as 10 tons of coke is needed per hour, this would practically correspond to 1 ton of coke for each ton of iron produced.

One ton of coke is equivalent to 1,000 kg., and as 1.18 kg. of coke is equivalent to 1 kg. of pure carbon, 1 ton of coke is equivalent to practically 850 kg. of pure carbon. If 4,153 kilogram-calories, as estimated by Yngstrom, be taken as the number of calories that are given off when 1 kg. of carbon combines with oxygen, forming a mixture that contains 30 per cent (by volume) of CO_2 ($0.3 \times 8,080 + 0.7 \times 2,470 = 4,153$), the heat necessary for producing 1 ton of pig iron in the blast furnace is $850 \times 4,153 = 3,530,050$ calories. As 2,121,284 calories represents the amount of heat that is necessary to produce 1 ton of pig in the electric furnace, there is a difference of 1,408,766 calories between the heat necessary for producing 1 ton of pig iron in the blast furnace and that required for producing 1 ton of pig iron in the electric furnace.

PROBLEMS IN THE ELECTRIC SMELTING OF IRON ORES.

Having thus traced the history and evolution of the electric pig-iron furnace up to the present time, and having stated the fundamental chemical principles upon which the reduction of iron is based, the authors will now briefly consider some of the difficulties that had to be overcome in order to bring the Swedish type of furnace up to its present stage of development and some of the problems that yet remain to be solved.

EARLY DIFFICULTIES.

An inspection of Harmet's drawings, and of the drawings of practically all others who gave their attention to the development of an electric furnace for the reduction of iron ores, shows that the first idea was to construct a shaft similar to a blast-furnace shaft, and then to substitute electrodes for tuyères. This seemed feasible, but in practice the furnace wall in the neighborhood of the electrodes proved to be short lived. Water cooling did not obviate this difficulty, but rather tended to increase it, due to jackets burning out and to other reasons. As was early observed by Héroult, the proper way to maintain the walls of an electric furnace crucible is to remove the electrodes as far as possible from the side walls. In the development of the furnace in California and in Sweden it was found necessary not only to do this, but also to keep the charge as far as possible

from the roof of the crucible, as otherwise the roof was short lived owing to the intense local heat generated at the point where the electrodes enter the charge. In overcoming this most serious difficulty the present shape of the roof of the crucible and the manner in which the electrodes are introduced into the crucible have been evolved. As a further protection to the roof of the crucible, a part of the gases that were escaping from the top of the shaft, as previously stated, was returned to the crucible in order to cool the walls, as well as to assist in the reduction of the charge.

ELECTRODE PROBLEM.

Although the Swedish experimenters did not have so much trouble with electrodes, in California the difficulty of procuring suitable electrodes greatly interfered with the progress of the work. When experiments were begun in California the manufacturers of electrodes in this country had not previously been required to furnish electrodes of such large size, namely, about 20 inches square and about 72 inches long. Of those first furnished, that part of the electrode projecting into the crucible would either break off completely after it became heated, or else would spall off in large chunks and give trouble in operating the furnace. That others also encountered this difficulty is evident from the following quotation from a paper^a presented by W. R. Walker at the April, 1912, meeting of the American Iron and Steel Institute:

Our problems—mechanical, metallurgical, and otherwise—proved many, and our experience soon demonstrated that the conditions surrounding the successful operation of a large electric furnace were in many respects entirely different from those involved in the use of smaller units. In illustration the demands of a 15-ton electric furnace proved to be far in advance of the art of manufacturing electrodes. Our necessities represented a requirement that the electrode manufacturers of America and Europe had not been called upon to meet, and it took much time and money before there was finally accomplished the 20-inch round amorphous-carbon electrode that is now being used at South Chicago.

At the plant of the Noble Electric Steel Co. it was finally decided to try graphite electrodes. These worked satisfactorily in all respects except that on account of the angle at which it was necessary to insert them in the crucible they were subjected to a severe strain which caused them to break at the threaded joints. The electrode problem is, however, no longer a serious matter. As stated by Walker, the 20-inch round electrodes now in use at South Chicago give satisfaction, and the engineers at Trollhättan also report that the large carbon electrodes of about the same size as those used at South Chicago meet their requirements.

^a Walker, W. R., Electric furnace as a possible means of producing an improved quality of steel; *Metall. Chem. Eng.*, vol. 10, 1912, p. 371.

UNSOLVED PROBLEMS.

In connection with the unsolved problems in the electric smelting of iron ores it may be well to take up first a discussion of the results obtained at Trollhättan and in California.

A most careful record was kept of the work done at Trollhättan during the period November 15, 1910, to April, 1911, and the data thus obtained were incorporated in a report submitted to Jern-Kontoret by two of its engineers—J. A. Leffler and E. Odelberg. The essential parts of the report have been translated by Dellwik.^a This report has been ably discussed by Robertson^b in a paper that he presented before the Toronto meeting of the American Electrochemical Society, September, 1911. The work at Trollhättan is also the subject of a paper by Frick.^c In this paper Frick also discusses the results obtained in California. Some of the points brought out in his paper are discussed below.

VOLUME OF SHAFT AS COMPARED WITH CAPACITY OF FURNACE.

Theoretically, the volume of the shaft of an electric furnace should be great enough to permit the practically complete reduction of the ore before it enters the crucible. In other words, the ideal condition in electric reduction furnace work would be to have the charge in such a condition as to require only melting by the time it comes into proximity with the electrodes. So far the degree of reduction that has taken place in the shaft of the electric furnace has varied all the way from nothing up to a considerable proportion less than complete reduction. Granted that the size of the shaft at Trollhättan has been properly calculated, and that the ratio of the volume of charge in 24 hours to the volume of the furnace should be 1:55, then in order to insure the best economical working conditions of the furnace, the reduction of the charge in the shaft must take place in the proper manner.

REDUCTION OF ORE IN SHAFT.

In an ordinary blast furnace the weight of the gases produced exceeds the weight of the charge by 30 to 50 per cent, whereas in electric-furnace reduction the gases evolved amount to only about 40 per cent, by weight, of the charge. In other words, in the blast furnace three to four times as much gas is given off in the production of 1 ton of iron as is given off in the production of 1 ton of iron in the electric furnace. Moreover, the temperature of the gas as it leaves the vicinity of the tuyères may be as high as 1,600° C., whereas the

^a Dellwik, C., *Electric iron smelting at Trollhättan, Sweden*: Iron and Coal Trades Rev., vol. 82, June 9, 1911, pp. 957-962.

^b Robertson, T. D., *Recent progress in electrical iron smelting in Sweden*: Trans. Am. Electrochem. Soc., vol. 20, 1911, p. 375.

^c Frick, Otto, *Electric reduction of iron ores with special reference to results obtained in Electro-Metals furnace at Trollhättan, Sweden, and Noble furnace at Heroult, Cal.*: Metall. Chem. Eng., vol. 9, Dec., 1911, p. 631.

highest temperature stated to have been attained at the lower end of the stack in the work at Trollhättan was 965° . Granting that the temperature of the gas as it enters the stack may be even $1,000^{\circ}$, inasmuch as the weight of the gases produced is only about 40 per cent of the weight of the charge, the charge would not be heated to more than about 350° C., as pointed out by Frick, owing to the fact that the specific heat of the gas and that of the charge are about the same. For this reason complete reduction in the shaft can not be accomplished solely by the heat from the gases generated in the regular manner. In other words, the shaft acts merely as a preheater, most of the reduction taking place in the crucible by means of solid carbon.

Thus it is seen that an electric furnace may be operated in one of two ways: The ore may be reduced in the crucible by means of solid carbon, with no attempt at reduction in the stack, as is done in Cal-

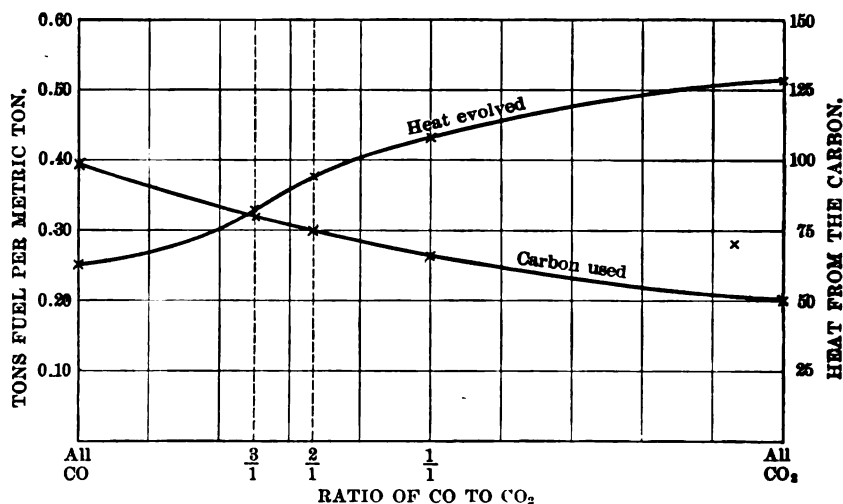


FIGURE 14.—Curves showing variation of fuel required and of heat evolved.

ifornia at the present time by the Noble Electric Steel Co., or there may be part or complete reduction in the stack, this being the principle on which the Swedish furnaces are operated.

As is well known, a method that may seem best theoretically is not always best in practice. In this instance it is not possible to judge by results, as sufficient data are not available, and so the matter can be viewed only theoretically. In the reduction of any given iron ore, a definite amount of oxygen must be removed for every ton of iron obtained. Therefore it would theoretically be best to remove all the oxygen in the form of CO. However, as the oxidizing action of CO₂ gas varies with different temperatures, such removal is not possible, as experiments show that the ratio of CO₂ to CO should not be greater than 1:1, whereas 1:2 is the determined ratio in which they are usually present in blast-furnace gases. The curves in figure 14,

the work of Prof. Richards, show that in electric-furnace work the greater the proportion of CO_2 in the gas the less is the amount of carbon required to remove the oxygen from the ore. Figure 15 (also prepared by Richards) shows that the consumption of electrical energy is proportional to the production of CO. Therefore, judging from deductions made from the curves, the efficiency of the electric furnace will be increased as reduction is effected by the gases in the shaft, and the most desirable ratio of CO_2 to CO in the escaping gases is 1:1. If this be granted, it would seem that reduction in the shaft is necessary, and the problem becomes how to effect the most satisfactory reduction in the shaft.

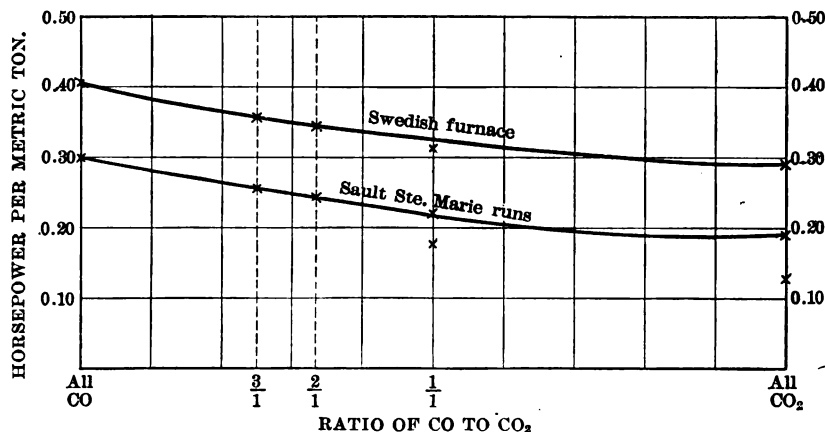


FIGURE 15.—Curves showing variation of power required according to the composition of the waste gases.

The essential factors in accomplishing reduction in the shaft are temperature, time, and the proportion of CO in the gas.

As has just been pointed out, the volume of the gas generated by the reduction of the oxides is not sufficient to maintain a reduction temperature in the stack. As a remedy for this difficulty the present system of gas circulation was adopted. The system is briefly discussed below.

GAS CIRCULATION.

As has been pointed out by Frick,^a the idea of using circulating gas in the electric furnace was conceived by Harmet and incorporated by him in his papers^b on the reduction of iron ores in the electric furnace.

^a Frick, Otto, Electric reduction of iron ores, with special reference to results obtained in Electro-Metals furnace at Trollhättan, Sweden, and Noble furnace at Heroult, Cal.: Metall. Chem. Eng., vol. 9, 1911, p. 631.

^b Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in Europe: Mines Branch, Department of the Interior, Canada, 1904, pp. 124, 139.

METHOD OF UTILIZATION.

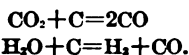
At present gas circulation is used in the Swedish type of furnace in order to cool the superheated brickwork of the crucible and to carry up through the stack a volume of gas that is large enough and hot enough to bring about reduction in the stack.

This method has been the subject of a great deal of discussion.

OBJECTIONS.

The following are the principal objections to the circulation of gas in the electric furnace:

(1) The moisture of the charge, the CO_2 of the flux, and the CO_2 gas naturally produced in the furnace are returned in a large part to the crucible and react on the unconsumed carbon by the reactions—



(2) The reactions noted materially cool the smelting zone of the furnace, which is its most vital working part.

(3) The system is cumbersome and expensive, and soon reaches the maximum of useful effect.

In order to overcome the objections mentioned it has been proposed to keep the ore in the shaft at a low red heat, and then to allow the CO gas produced by reduction in the crucible to rise slowly up through the charge, thus giving it the best opportunity of producing the maximum amount of CO_2 . In the circulation of gas, as has been pointed out by Richards,^a it is not the increased volume of the circulating gas that performs the reduction, as it is evident that when the amount of gas passing through the furnace is increased two or three times its velocity is also increased to that extent, and therefore its contact with the ore is only one-half to one-third as long. Such being the case, the only value of circulating a part of the gas escaping from the top of the shaft, so far as the problem of reduction is concerned, is that the charge in the shaft is kept at the temperature necessary for reduction, but, on the other hand, the CO_2 returned to the crucible is reduced by C to CO, and thus one aim of electric-furnace reduction is defeated, namely, the removal of the oxygen from the ore with a minimum amount of carbon.

Nevertheless there are objections to the methods that have been proposed for obviating the difficulties connected with the present method of gas circulation which may be briefly stated as follows:

(1) It would probably be difficult to heat successfully the charge in the stack by means of surface electrodes embedded in the walls of the shaft.

(2) As at a low red heat (say 480°C.) reduction takes place slowly, the volume in the shaft would have to be large in order to

^a Richards, J. W., Gas circulation in electrical reduction furnaces: *Trans. Am. Electrochem. Soc.*, vol. 21, 1912, p. 403.

give the charge time enough to be reduced in passing through the shaft.

(3) The use of burnt limestone, where shaft reduction is attempted, would introduce an unnecessarily large quantity of fine material into the charge.

(4) The water-cooled plates in the arch of the crucible, although useful, would not of themselves form a satisfactory substitute for the introduction of cold gas beneath the roof of the crucible.

As has been stated by Leffler,* one of the engineers of the Jern-Kontoret, gas circulation would be abandoned for a more simple and practicable method if such should be discovered.

However, the use of gas circulation at Trollhättan seems to be necessary, at least with the Swedish type of furnace. If this be true, efforts to improve the system ought to be directed somewhat along the following lines:

As is now done at Trollhättan, that part of the gas from the top of the shaft that is to be circulated could be drawn through a "Cottrell system" for the removal of the solid particles of ore and flux, by means of the circulating fan, and forced through a regenerator filled with coke heated electrically to such a temperature as to bring about the reduction of the CO_2 to CO . The gas at the same time would be heated, but not to a temperature too high to absorb heat as it entered the crucible, and would thus pass up into the shaft at such a temperature as to readily effect reduction. Part of the excess waste gases could then be burned in a jacket around the shaft of the furnace so as to prevent radiation losses; the waste gases from the jacket could be passed to a preheater and completely burned, whereby the charge, minus the reducing agent, would be dried and raised to a temperature high enough to admit of the ore being reduced as soon as or soon after it was introduced into the stack.

The obvious advantages of such a system would be the doing away with the introduction of water vapor into the gas, as is done by the present system of scrubbing the gas; the removal of CO_2 from the gas; and the raising of the temperature of the circulating gas.

Although the system as above outlined is just as cumbersome and expensive as the present method, it would probably be more effective.

OTHER METHODS SUGGESTED.

Some of the methods suggested as improvements are:

(1) *Calcining the limestone outside of the furnace.*—This subject is discussed in a later section dealing with the comparison of the Swedish and Californian types of furnaces.

* Leffler, J. A., In discussion of Richards's paper on "Gas Circulation in Electrical Reduction Furnaces": Trans. Am. Electrochem. Soc., vol. 21, 1912, p. 412.

(2) *Preheating the ore.*—Judging by experience gained in California, the writer believes that the preheating of the charge is beneficial, for the following reasons: (a) It dries the charge and thus permits a more accurate weighing of the same, which is especially important in the electric reduction furnaces; (b) the initial temperature of the charge on entering the stack is thus sufficiently high to permit its being reduced immediately.

(3) *Smelting of fine ores.*—The authors are privileged to quote from a communication from Electro-Metals, Ltd., in regard to this matter, as follows:

As was explained in our first publication on this subject, the object of this plant [at Trollhättan] was to determine the relative merits of electric smelting as compared with ordinary blast-furnace smelting. For this reason the work has been carried out under widely varying conditions and with different kinds of ore and fuel. In consequence the results are by no means uniform and scarcely suitable for conclusions based on the average figures.

One object of some importance in Sweden was to determine the proportion of ore concentrates which could be used. The results prove that a large proportion of concentrates is detrimental to smooth running and good results.

This is readily understood from the fact that only about one-third as much charcoal is used as in the blast furnace, and concentrates therefore have an increased tendency to choke the passage of the gas.

As will be seen from the above, it is not feasible to use a large proportion of fine material in making up the charge for the electric furnace. This limitation, however, does not prevent the smelting of fine ores, for they may be sintered.

(4) *Sintering.*—The advantages derived from sintering in electric-furnace work would be as follows: (a) The fine material would be caked in lumps, permitting free passage of the gases up through the charge in the shaft, and as the lumps would be porous they would be readily reduced, and (b) the fine ore would be preheated; that is, the hot sintered ore could be charged directly into the shaft at reduction temperature.

(5) *Increasing the size of the unit.*—The electric reduction furnaces now in operation vary in size as regards their horsepower from 1,500 up to 3,500. The largest yet designed requires 7,500 kilowatts and is at a plant in Sweden. As can be readily understood, it is important that the size of the unit be made as large as possible, and it is quite probable that larger furnaces will be built.

(6) *Increasing the general efficiency of the furnace.*—Improvements will probably be along the following lines: (a) The utilization of the waste gases; (b) the securing of a high-power factor; (c) the correction of induction losses; and (d) the perfecting of the single-phase furnace as recommended by Catani.^a

^a Catani, R., Large electric furnaces in the electrometallurgy of iron and steel: *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 168.

STATUS OF THE IRON INDUSTRY IN THE WESTERN STATES.

At the present time the only important iron-reduction plant west of the Mississippi River is at Pueblo, Colo. The plant has six stacks, twelve 50-ton open-hearth furnaces, two 15-ton Bessemer converters, four 400-ton furnaces, and two 250-ton furnaces. That there are no other large iron-reduction plants in the West is due largely to the fact that there is lacking in that part of the country, especially in California, the grade of coal necessary for making a suitable metallurgical coke. Then, too, although there are several known large iron-ore deposits scattered throughout the Western States, not much attention has in the past been paid to this class of deposits.

For nearly half a century after the discovery of gold in California the prospector along the Pacific coast looked for gold-bearing rock only. In recent years the discovery of valuable deposits of copper in California gave the prospector the copper craze, so to speak. In prospecting he looked for iron only as an indication of the presence of copper; that is, if he came across an iron deposit he looked upon it as a gossan or capping, and at once examined it for gold and copper, as his experience had taught him that gold is frequently found near the surface in rock of similar appearance and copper at a greater depth. It is well known that there are many deposits that yield oxidized ores above water level and sulphides below, so it is easy to understand why the prospector should have mistaken a true iron deposit for a deposit of the nature just mentioned. Even such iron ores as those in Shasta County were never seriously considered as such by prospectors. The miners in that part of the country commonly believe that iron indicates the presence of copper, and they insist that copper will be found below the iron.

IRON-ORE DEPOSITS ON THE PACIFIC COAST.

As above stated, it has long been known that there are large deposits of iron ore in Nevada, Arizona, and California, especially in southern California. As has been pointed out by Jones,^a most of the present known deposits of the Southwest have been discovered in places where there has been more or less erosion. Owing to the erosion, faces of ore several hundred feet in height, and of as great or greater width, are shown in certain intersecting gulches. Intrusions in limestone beds are common in the desert regions of Arizona, western Nevada, and California, and, as Jones states, there is in these regions scarcely a range of this nature that does not show some ledges of iron or float ore, and he ventures the assertion that syste-

^a Jones, C. C., Iron ores of the southwest: Paper read before the American Mining Congress, September, 1910; published in *Min. and Min.*, vol. 31, April, 1911, p. 574.

matic prospecting and mining would uncover as great bodies of ore as have been already accidentally exposed by erosion.

For some time past the railway and oil interests of California have been systematically acquiring iron-ore holdings. For example, Jones^a states that in 1911 the Union Oil Co., through its subsidiary company, the California Industrial Co., was reported to hold an aggregate proven tonnage of 300,000,000 tons, one-third being in California and two-thirds in Lower California, Mexico. This company has acquired these properties for the reason that they believe the problem of using oil as a reducing agent will ultimately be solved, and thus permit the establishment of an iron and steel industry in that section of the country. The Southern Pacific Railroad Co., operating under the name of the Iron Chief Mining Co., controls several deposits throughout the State, one of which is in Riverside County, some 140 miles from Los Angeles. Jones^b states that it has some 30,000,000 tons of proven ore of such a nature that it can be mined cheaply. The average analysis of the ore is claimed to show 64 per cent iron and phosphorus within the Bessemer limit.

In addition to these deposits Jones also mentioned the following: At Scotts Siding, 190 miles east of Los Angeles, where 10,000,000 tons of ore have been blocked out, there being three times that quantity of ore on the property; in the Providence Mountains, 236 miles from Los Angeles, is a deposit of 5,000,000 tons of soft hematite, of Bessemer quality, that can be mined with a steam shovel; 12 miles west of Silver Lake Station and 230 miles east of Los Angeles is a deposit owned by the Colorado Fuel & Iron Co. that it states shows over 13,000,000 tons of ore that can be mined with steam shovels. In addition to the deposits named there are the Minarets deposits in Madera County, which are at present inaccessible, being about 80 miles from the railroad, but it is thought that a railroad line will soon be built into this district, thus making them accessible both to San Francisco and Los Angeles. It is stated that this is one of the largest deposits of iron ore to be found anywhere in the United States. As a result of his investigation Jones is of the opinion that there is 200,000,000 tons of available high-grade ore in southern California alone within 300 miles of Los Angeles, and that the ore can be laid down at Los Angeles at a cost of \$3.50 to \$4 per ton. If these figures are correct, there is a basis for the hope that an iron industry will be established on the Pacific coast, provided there can be found a commercially feasible process that will permit the utilization for metallurgical purposes of those fuels that are available in that part of the country.

^a Jones, C. C., *Op. cit.*, p. 53.

^b Jones, C. C., *Idem.*

DEVELOPMENT OF THE ELECTRIC REDUCTION FURNACE IN CALIFORNIA.

ORE DEPOSIT OF SHASTA IRON CO.

For 30 years or more a company known as the Shasta Iron Co. has owned a deposit of iron ore situated about 7 miles from the mouth of the Pitt River, in Shasta County, Cal. This ore is a high-grade magnetite. An average of a large number of analyses gives the following percentages:

Average of analyses of magnetite from Shasta County, Cal.

	Per cent.
Fe ₃ O ₄	89.4
Fe ₂ O ₃	7.3
Fe	69.90
MgO	.10
MnO	.18
SiO ₂	2.40
P	.011
S	.009

The ore is found near a contact between limestone and diorite. The ore body is markedly uniform, a series of 20 samples taken by the writer over a cut 60 by 40 feet giving the following iron content:

	Per cent.
Maximum	70.5
Minimum	68.8
Mean	69.7

The limestone is also of an excellent quality; an average analysis is as follows:

Average of analyses of limestone from Shasta County, Cal.

	Per cent.
SiO ₂	1.2
Al ₂ O ₃	.5
MgO	1.1
CaO	53.8
FeO	.2

The percentage of CaO is equivalent to about 98 per cent calcium carbonate.

Although the Shasta Iron Co. had previously planned to make pig iron from this ore, nothing definite was done until the summer of 1906. At this time the possibility of smelting the ore by electricity was brought to the attention of H. H. Noble, president of the Northern California Power Co. Owing to Héroult's connection with the experimental work that had been done at Sault Ste. Marie, he was commissioned to construct a plant for the purpose of treating the ores above mentioned. The place selected was at a point on Pitt River, near the mines. The place was named Heroult, and is still known by that name.

FIRST EXPERIMENTAL ELECTRIC REDUCTION FURNACE.

In July, 1907, the first furnace having been completed, experimental work was begun. This furnace was a 1,500-kilowatt, three-phase furnace of the resistance type. It soon presented mechanical difficulties that made its commercial use impracticable, and so it was closed. This furnace is shown in figure 16. In construction the crucible resembled a long, elliptical-shaped iron box. The bottom was made of heavy cast-iron plates, into which had been cast cast-iron lugs or pins, and upon the iron plates was tamped a bottom made of carbon paste. The bottom plates were connected with the bus

bars in such a manner as to form a neutral in the three-phase system.

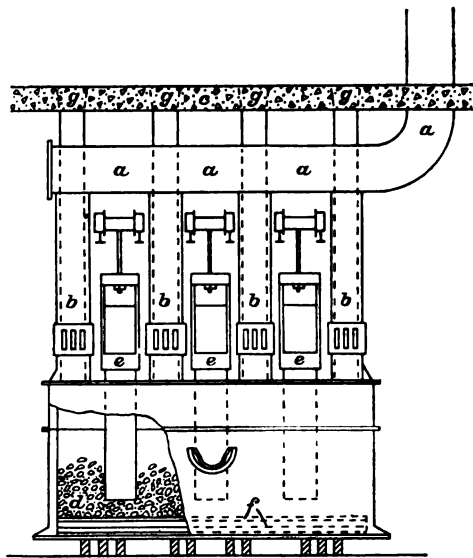


FIGURE 16.—Héroult 1,500-kilowatt, three-phase reduction furnace.

The electrodes passed through the roof, as shown in the figure. They were suspended from copper holders which were water jacketed. The charge was fed into the crucible through the pipes, *b*, and the gases from the crucible passed up around *b* into *a* to preheat the charge in *b* by burning the gases around *b*, air being admitted for that purpose through slots in the outer pipe just above the roof of the crucible. The pipes, however, proved too small, and the charge in them be-

came so hot as to hang, and so they had to be discarded. It was also impossible to maintain a roof over the crucible, and so the furnace was finally operated as an open-top furnace. Several tons of excellent foundry iron was made in this furnace, and later it was used for making ferrosilicon, but was finally closed and removed in order to make room for the type of furnace subsequently adopted.

SECOND AND THIRD EXPERIMENTAL FURNACES.

After the closing of this furnace experimental work was conducted in a 160-kilowatt single-phase furnace, and, using the results obtained in this furnace and its method of operation as a basis, a second 1,500-kilowatt three-phase furnace was designed and built. This

furnace is shown in figure 17. Above the crucible was a superposed shaft resembling that of an ordinary blast furnace.

In the operation of the furnace the ore, mixed with the proper proportion of flux, was fed into a preheater, *b*, that was heated by waste gases from the combustion chamber *k* at the top of the shaft. The gases entered the base of the preheater through a flue, *e*, from an annular chamber surrounding the top of the shaft and communicating with the chamber *k* through openings.

A mechanically operated scale car, *g*, moved on a circular track around the top of the stack and received first a weighed charge of ore and flux from the preheater *b* and then a weighed charge of charcoal from the charcoal hopper *c*, delivering the charges alternately into the furnace.

The charge was kept at about the height of the irregular line shown in the shaft. Above the charge were small openings in the shaft with suitable valves for admitting the requisite amount of air to burn the gases resulting from the reduction of the ore in the lower part of the stack, thus still further heating the charge; as above stated, the waste gases were then passed up through the preheater *b*, thus preheating and drying the ore before it was charged into the furnace.

The electrodes, six in number, were spaced equidistant around the furnace, so that the current passing between them melted the charge, and the molten metal and slag were collected in the crucible, from which they were drawn as in ordinary blast-furnace work. The tap holes were arranged at different levels, so that the crucible might be partly or completely emptied, as desired.

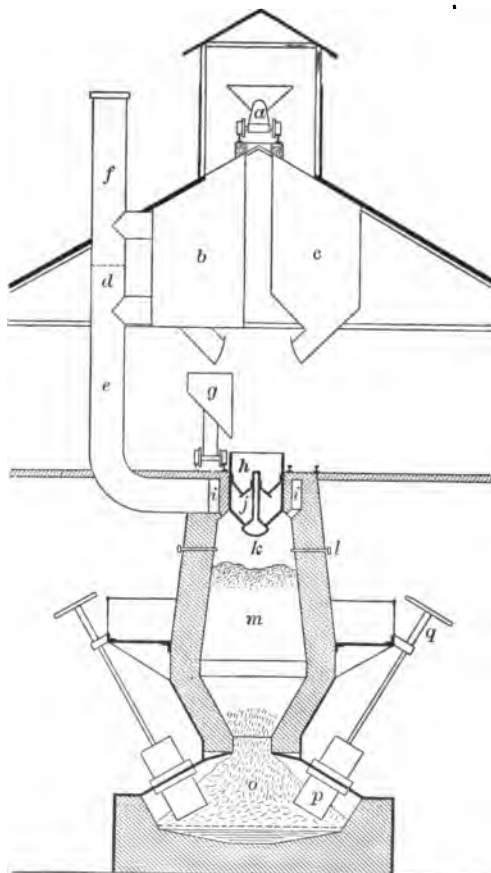


FIGURE 17.—Second 1,500-kilowatt, three-phase reduction furnace built at Heroult, Cal.

The electrical equipment consisted of three oil-insulated, water-cooled transformers, having a rated capacity of 750 kilowatts each, 60 cycles, 2,200 volts primary, and a secondary range of 35 to 75 volts, constant output. The secondary current varied from 10,000 to 21,400 amperes.

The range in voltage was controlled by a dial switch in the primary circuit, giving steps of about 3 volts in the secondary. The efficiency was 98.6 per cent at 75 volts on the secondary, guaranteed for 25 per cent overload for two hours. The rise in temperature at full load, by test, was 35° C.

The generation of energy in the furnace was, therefore, controlled externally without movement of the electrodes, whose position was changed only to accommodate their wear in the crucible.

Many experimental runs were conducted in this furnace, and many changes and alterations were made in the construction and shape of the roof of the crucible, the manner of connecting the electrodes, the height and volume of the stack, the method of pre-heating the charge, the circulation of gases through the crucible and up through the shaft, and the like.

Although several hundred tons of splendid foundry iron was made in this furnace, much of the product was white iron, for the production of which this type of furnace seems best suited, as has also been ascertained at Trollhättan.

PRESENT TYPE OF FURNACE.

In the spring of 1911, owing to electrode troubles and to the irregular composition of the iron produced, the company decided to abandon the type of furnace that had been developed and to attempt to evolve a furnace in which iron could be produced that would better meet the requirements of the Pacific coast foundries. In this connection it may be stated that scrap iron is largely employed in that part of the country for making all grades of castings and that the sulphur, phosphorus, and combined-carbon content of many of the castings is so high as to impart very undesirable qualities to the finished product. For this reason the Pacific coast foundrymen have been in need of an iron high in graphitic carbon and silicon and low in other impurities. The Noble Electric Steel Co. states that it is able to produce such a grade of iron in the type of furnace now being used, and that the following is a fair average analysis of the pig iron that it is producing:

	Per cent.
Si.....	3.64
S.....	.00
P.....	.02
Combined carbon.....	.00
Total carbon.....	3.58

The details of the construction of this furnace have been furnished by the company, and are presented below, being taken chiefly from a recent article by Van Norden:^a

THE PLANT.

The furnace is housed in a heavy steel-frame building, sheeted with corrugated galvanized iron. It is 120 feet long and 75 feet wide, and about 60 feet high. The interior is divided into two sections or bays by a line of columns. In the south section is the pouring floor, on which the pigs are cast. Throughout this bay is operated a 50-foot traveling crane. This is electrically operated and has one 20-ton and one 5-ton hoist. A lifting magnet, having a capacity of 2,000 pounds, is used in picking up the pigs. It will lift about 1,000 pounds of hot pigs.

The north bay contains the electric furnaces, their transformers, and the necessary controlling equipment.

THE FURNACE.

The crucible is contained in a steel shell 27 feet long, 13 feet wide, and 12 feet high. The crucible is rectangular, with a sloping bottom to facilitate the flow of the molten bath, when the furnace is tapped. The tap hole is in the center of the front of the crucible. The roof of the furnace is arched, and is penetrated by five stacks 24 inches in diameter and extending 15 feet above the roof of the crucible. In the four spaces between the stacks at the center of the dividing arches are inserted graphite electrodes. The electrode jackets and the arched roof are water cooled. The stacks are used only for charging, no reduction being attempted in them. A charging platform is built along the top of the stacks and supports a track which leads to the mixing platform. The loaded car is run along this track and the charge dumped directly into the stack. Except when receiving a charge, the stacks are closed with tight-fitting caps.

THE ELECTRODES.

Graphite electrodes are used. They are cylindrical in form, 12 inches in diameter, and 4 feet long. The upper end has a tapered male-threaded nipple, and the lower end has a corresponding socket with a female thread. As the electrode is fed into the charge a new one may be fastened to it, making a continuous feed.

An electrode in continuous operation lasts 30 days. Occasionally an electrode breaks in the furnace. Such breakage was formerly a serious matter and caused considerable delay and annoyance in the operation of the furnace, but this difficulty has now been practically obviated.

^a Van Norden, R. W., Electric iron smelting at Heroult, Cal.: Jour. Elec. Power and Gas, Nov. 23, 1912.

THE TRANSFORMERS.

In the rear of the furnace, and as close thereto as is possible, are the three service transformers which supply three-phase current at 40 to 80 volts to the electrodes. These transformers are oil immersed and water cooled and have a capacity of 750 kilowatts each. The low-tension connections to the electrodes consists of eight pieces of flat copper bar, three-eighths of an inch thick and 5 inches wide, bolted together. On the 2,400-volt primary side there are brought out eight current taps for voltage regulation. These lead to an oil-immersed switch group, each unit of which is operated by a solenoid. This arrangement, with autotransformer compensator, gives 15 steps for voltage variation.

ELECTRIC CONTROL.

Electric control is through a switchboard, there being a panel for each furnace. As the current and power factor in each phase must be under observation at all times during operation, separate meters are installed in each phase. The requirements for one panel are 3 ammeters, 3 voltmeters, 3 wattmeters, 3 power-factor meters, and 3 recording wattmeters. The meters are mounted across the panel in rows of three each. Under the first four sets named are three hand-wheels to control the voltage variation, and under these three switches that control the entire load, and under the latter are the recording wattmeters.

For operating the voltage control and the main circuit breakers there is a $7\frac{1}{2}$ -kilowatt motor-generator set, comprising a 125-volt direct-current generator, directly connected to a 10-horsepower induction motor. This set has a small panel board mounting a circuit breaker, ammeter, voltmeter, and two single-pole knife switches. In the event that line voltage should fall, or if for any other cause the direct-current supply should become deranged, there is a storage-battery set, having a capacity of $7\frac{1}{2}$ kilowatts, which may be instantly switched in, and thus prevent the furnaces from cutting out in the case of low voltage.

OPERATION.

The operation of the furnace is continuous. After a period of eight hours the hearth contains a full bath of molten metal, and therefore the metal is tapped three times each day. Charging is done at regular intervals, and the current is never shut off during operation. During the period of smelting the change in electrical conditions is interesting. At the beginning of the charge the power factor is almost unity. This gradually lowers as melting continues, until with a full bath of metal a power factor of 65 per cent is reached. If coke is used in place of charcoal, or if a mixture is employed, a dif-

ferent set of power-factor conditions exists. By studying these conditions it is possible to know the exact condition of the charge by looking at the meters. The load is, of course, a function of the voltage, and with half voltage the load will drop one-quarter.

In charging, the ore cars are run on the mixing floor, which is level with the charging floor. Charcoal and lime and quartz for flux are brought in on the lower floor in cars, hoisted on an electrically operated elevator to the mixing floor, and are dumped into their respective bins. The mixing is done in a car that is run on platform scales. The charge is placed in layers, the proportions depending on the tests made in the laboratory by the chemist. Following is a representative charge: Five hundred pounds of iron ore (magnetite); 135 to 150 pounds of charcoal; $3\frac{1}{2}$ pounds of lime (well burned); $12\frac{1}{2}$ pounds of quartz.

USE OF CHARCOAL AS A REDUCING AGENT.

Although, as explained later, the use of the electric furnace in the production of pig iron permits the manufacture of 3 tons of pig iron from 1 ton of coke or charcoal, the cost of a suitable reducing agent is still an important factor, especially in California. Charcoal is used at the plant of the Noble Electric Steel Co., and its economical manufacture in retorts so constructed as to save the distillation products has been almost as much of a problem as the production of the iron itself. Pit charcoal was used at first. A battery of beehive charcoal ovens, each holding 60 cords of wood, was then constructed. Later it was decided to use by-product retorts, and a vertical retort system was accordingly built. The retorts were arranged in two batteries of four each, each retort being a vertical cylinder 6 feet in diameter and 16 feet high. The volatile products were led from the bottom of the retorts to condensers. Altogether, wood distillation and the attendant recovery and working up of the by-products has not seemed to be a profitable enterprise on the Pacific coast. Soft woods are used for the most part, and these are generally lean in acetic acid and wood alcohol, although fairly rich in tarry products.

A splendid grade of charcoal was made at the plant of the Noble Electric Steel Co. in the retorts above mentioned, but the system proved to be rather cumbersome, and the time necessary for retorting was excessive. Moreover, owing to the manner in which the retorts were constructed, all the products resulting from distillation of the wood had to pass down through the hottest part of the retort. As a result the volatile products were broken up by heat, undesirable products were thus formed, and an inferior grade of by-products was produced. Consequently the company has decided to replace this system with what is known as the Yost system. The retort of the Yost system consists of a horizontal steel cylinder, about $5\frac{1}{2}$ feet

in diameter and about 20 feet long, mounted in brickwork. One end of the retort is closed and the other is provided with doors. The wood is loaded on a car, and the car is run into the retort, which is then closed and sealed. The volatile products resulting from the heating of the wood are conducted to the condensers by means of a copper pipe; the liquid tar formed in like manner is drained from the bottom of the retort into a tank. Owing to its simplicity this type of retort is claimed to be much less expensive than the type formerly used, and because of the fact that the wood will not have to be handled as much as formerly the cost of producing charcoal is claimed to be much less.

USE OF CRUDE PETROLEUM AS A REDUCING AGENT.

As is well known, California is favored with a seemingly abundant supply of petroleum. Due to the scarcity of suitable reducing agents such as are at present used in metallurgical work attempts to use crude oil for this purpose have from time to time been made. It is quite possible that a successful process of this kind will be devised. The Bureau of Mines has been conducting investigations along this line and expects to publish the results in the near future. As soon as a commercially successful process of this nature has been devised, it is quite likely that the electric smelting of iron ores on the Pacific coast will receive an impetus, for the use of crude petroleum will not only permit the introduction of electric smelting in those districts where its use, except at prohibitive prices, is not now possible owing to the lack of suitable reducing agents, but should also cheapen the cost per ton of metal produced as compared with the cost when charcoal is used.

COMPARISON OF THE CALIFORNIA AND SWEDISH FURNACES AND PROCESSES.

As to the construction of the two types of furnaces, the difference is so apparent that no comment is necessary. As to the process, the California practice differs from the Swedish practice very decidedly in the following respects, namely: No attempt is made to reduce the ore in the stacks of the furnace; no artificial circulation of gases is employed; limestone calcined outside of the furnace is used. In other words, the California practice is just the reverse of the Swedish practice.

REDUCTION IN SHAFT.

As has previously been noted, the ideal condition in electric reduction furnace work would seem to be to have the charge in such a condition as to require only melting by the time it comes in proximity to the electrodes. The electric current would then be used to furnish only the additional heat necessary to bring about the melting

of the hot previously reduced oxide. Of course the heat for the reduction must come from some source, and its origin would be in the crucible and as a result of the heat developed during the melting of the charge, but the excess heat so developed can be transferred by the circulation of the gas to the charge in the shaft instead of being carried away by radiation and by circulating water.

CIRCULATION OF GASES.

As has also been mentioned on page 16, in the Swedish practice a part of the gases rising from the top of the shaft is returned to the crucible and made to pass through it and up through the charge in the shaft. This is done to cool the superheated brickwork of the roof of the crucible and to carry through the stack a volume of gas large enough and hot enough to bring about reduction in the stack.

As stated elsewhere, one of the most serious difficulties that had to be overcome in the development of the Swedish type of furnace was the destruction of the crucible resultant on local heating in the vicinity of the electrodes. Therefore, by returning to the crucible a part of the gases resulting from reduction, it is possible to prevent excessive local heating and at the same time to make use of the heat instead of wasting it. Although gas circulation has its objections, its advantages in the way of economy of operation would seem to outweigh its disadvantages. On the other hand, if reduction previous to smelting be considered the proper practice, this reduction will have to be made in the stack, and the present California practice will not find application except when warranted by the local conditions. As has been previously stated, the present demand in California is for a soft gray foundry iron. Although the furnace now used gives that product, such a construction and such a process are, of course, not absolutely necessary for the production of such an iron.

That the California type of furnace is better suited to the production of a soft gray foundry iron than is the Swedish type is probably due to the fact that reduction is performed solely by solid carbon, and the reduction of silica to silicon takes place at the same time as the reduction of iron oxide to metallic iron. The silicon is dissolved in the iron and by its presence causes the carbon to be precipitated as graphitic carbon, and there is thus obtained the soft gray iron desired. It is also quite probable that in a furnace having a rectangular crucible, such as has the California furnace, the temperature of the charge in the crucible as a whole is much higher than in a furnace of the Swedish type, and this higher temperature is more favorable to the reduction of silica to silicon. As before stated, the latter reaction is largely the controlling element in the production of soft gray foundry iron.

USE OF CALCINED LIMESTONE.

The use of calcined limestone has been repeatedly suggested, and theoretically seems desirable, but in operating a furnace of the Swedish type there are two objections to its use, namely, it increases the proportion of fine material in the charge and it makes the charge hang.

As to the former of these objections, the idea seems to be prevalent that there is no limit to the percentage of fine material that may be used in the electric reduction furnace. However, judging from observations of the smelting of black sands when the furnace consisted simply of a crucible and no shaft, some difficulty was experienced on account of the charge being made up entirely of fine material. In the Swedish practice the proportion of fine material that can be used in a charge was definitely determined in the work at Trollhättan, and it was found that a large proportion of fine material was detrimental to smooth running and good results; but Leffler, one of the engineers in charge of the work at Trollhättan, is of the opinion that calcined limestone may be advantageously added to the charge if no reduction is attempted in the shaft.

In view of the above facts, it would seem that the California practice as compared to the Swedish practice presents the following advantages: It permits the use of limestone calcined outside the furnace and it does not require the circulation of gases.

As to the circulation of gases and also as to reduction in the shaft, the California practice might perhaps prove more efficient if both were done. Although a very complete record of the working of the furnace at Trollhättan has been published, no such record of the operation of the furnace at Heroult is available, and so it is not at present possible to make a definite comparison of the two practices.

ELECTRIC FURNACE AS COMPARED TO THE BLAST FURNACE.

As has been mentioned in the section pertaining to the history and evolution of the electric pig-iron furnace, the electric furnace was not developed as a competitor of the blast furnace, but for the purpose of finding a furnace and a process that would be able to produce iron in those localities where blast-furnace practice was not feasible, or, as in Sweden, where the increasing cost of suitable fuel was becoming prohibitive to the existing practice of smelting in blast furnaces.

Broadly speaking, it may be said that the feasibility of smelting iron ores in an electric furnace depends upon the relative cost of either charcoal or coke and of electric power. As regards the latter, it must be cheap. As is well known, electric power at the present time can be developed more cheaply than it could when Capt. Stassano made

his experiments in 1898, and it will probably be produced still more cheaply in the future. However, the present purpose is not to speculate on this point but to state the facts as they are now understood. As has been previously mentioned, the average consumption of charcoal or coke in the electric furnace in the production of 1 ton of pig iron is about one-third of what it is in the blast furnace. Knesche^a has pointed out that when one takes into consideration the consumption of electrodes, the saving is practically 0.7 ton of coke or charcoal per ton of iron produced.

HYPOTHETICAL CASE.

To make the matter more definite a hypothetical case is assumed, as follows: Coke is assumed to cost \$6 a ton and a kilowatt-year to cost \$16. From the report of the work done at Trollhättan it is learned that, on an average, 2,391 kilowatt-hours is required to produce 1 ton of pig iron. Hence, the cost for power per ton of pig iron produced would be \$4.37, to which must be added the cost of 0.3 ton of coke and likewise the cost of 11.6 pounds of electrode, or an itemized cost as follows:

2,400 kilowatt-hours, at \$16 per kilowatt-year-----	\$4.37
0.3 ton of coke, at \$6-----	1.80
11.6 pounds of electrode, at \$0.06-----	.70
	6.87

As 1 ton of iron can be produced in a blast furnace from 1 ton of coke, the respective costs in the blast furnace and in the electric furnace would be \$6 and \$6.87. Hence, if considered on this basis alone, the electric furnace, with coke at \$6 per ton and power at \$16 per kilowatt-year, could not compete with the blast furnace. There are, however, other items to be taken into consideration, such as initial cost of plant, quality of iron produced, and especially the efficiency of the furnace, for, as has been pointed out by Ashcroft,^b "the efficiency of many nonelectrical furnaces is barely 10 per cent of the theoretical, and very few will exceed 25 per cent, whereas the efficiency of electrical appliances sometimes reaches 75 per cent and is often 50 per cent."

Hence, with no greater difference between the respective costs than is shown above, a careful investigation of all items that enter into the total cost of production of a ton of pig iron, as well as the nature of the ore to be treated and the grade of product desired, might disclose that it would be more profitable to produce pig iron in the

^a Knesche, J. A., *Electric smelting of ore in the United States: Iron Trade Rev.*, Jan. 5, 1911, p. 65.

^b Ashcroft, A. E., *The influence of cheap electricity on electrolytic and electrothermal industry: Trans. Faraday Soc.*, vol. 4, 1908, pp. 134-142.

electric furnace than in the blast furnace. There are, however, few localities where coke costs only \$6 a ton and where the cost of an electrical kilowatt-year is \$16. In more localities the cost of coke is \$12 to \$14, and with coke at these figures the advantage is decidedly with the electric furnace, with power at \$16 a kilowatt-year.

COST OF POWER.

In those electric-furnace iron plants that are operated at the present time only hydroelectric power is used. The cost of producing power for electric furnace work must, of course, vary with local conditions and hence depends on the initial cost per kilowatt of installation. In general there are few localities where the electric smelting of iron ores would be feasible with the electrical energy costing more than \$20 to \$30 per kilowatt-year.

USE OF ELECTRIC-FURNACE PIG IRON IN THE OPEN-HEARTH FURNACE.

The first metal produced at Trollhättan did not look much like pig iron and, moreover, as its analysis was different from that of the metal that had been used, doubt was expressed as to the possibility of using it in the production of steel. The first attempt to use the iron in the production of steel was made at Degerfors, where a basic furnace was used. The iron of the electric furnace was mixed with ordinary gray iron and a smaller quantity of scrap than usual. This mixture was made so as to insure a hot metal upon melting down. The results exceeded expectations, and on the third trial the charge was composed of only the electric-furnace metal and scrap. Much to the surprise of those operating the furnace, it was observed that the boiling which generally takes place immediately after the metal is melted was not so violent as usual. Tests were then made on the electric-furnace metal in an acid furnace. Although small quantities of the metal were at first used, it was soon ascertained that the charge could be composed of the ordinary proportions of 64 per cent pig iron and 36 per cent scrap.

Pig iron of normal composition was finally obtained as a product of the Trollhättan furnace, and when this iron was used for steel melting in the open hearth, together with the same proportion of scrap as had been used in the previous tests, a longer time was required to complete the refining of the metal, and it was necessary to add more ore than in the previous tests.

As a result of the tests at Degerfors the conclusion was reached that a metal could be produced in the electric reduction furnace that would be more suitable for steel making than ordinary pig iron.

In order to understand the reason for this conclusion, it may be well to note briefly some of the reactions that take place in the open hearth when treating pig iron of normal composition as compared to those which take place when treating pig steel.

In converting normal blast-furnace pig iron into steel in the open-hearth process, the pig iron must contain enough silicon to protect the iron from oxidation during the melting, and likewise enough silicon and manganese to free the iron after melting from the oxides and gases that were formed and taken up as a result of the action of the blast during the production of the pig iron in the blast furnace. The silicon and manganese that may be present in excess of this amount only prolong the time required to finish the heat.

During and immediately after the melting the metal boils violently, especially in an acid furnace if a low-silicon iron be used, even though it is not spongy. There is a second and less violent boiling when the metal is charged into the ladle and molds, especially in the case of hard steel. On the other hand, when electric-furnace "pig steel" is heated in the open-hearth furnace process, the second boil does not take place, and the regular boil commences immediately after the charge is melted and before the ore is added. Several theories have been advanced to account for such behavior of "pig steel" upon melting. One theory is that the metal is practically free from minute particles of oxides that are generally present in ordinary pig iron and that, as previously stated, are produced by the reoxidizing effect of the blast. Owing to the absence of the oxides, practically no reaction with the silicon and manganese present occurs at this time, and on this account the second boil in the open-hearth furnaces during and after the melting, when treating electric-furnace metal, does not occur, and it is possible to decarburize the charge at once by ore additions. The presence of silicon in such "pig steel" is therefore unnecessary during the first part of the open-hearth process.

As above stated, in the tests at Degerfors it was found that more time and more ore were required in the treatment of normal electric-furnace pig-iron than in the treatment of normal blast-furnace pig-iron. This is no doubt due to the fact that inasmuch as electric-furnace pig-iron of normal composition contains no oxides it is not necessary that silicon and manganese be present in the iron in order to take up oxides, and hence the removal of the silicon must be accomplished principally by means of ore, and, as pointed out by Odelberg,^a only after this is accomplished can decarburization proper begin.

In conclusion it may be said that "pig steel" made in the electric furnace is better suited to the production of open-hearth steel than is the normal blast-furnace pig iron and that normal pig iron made in

^a Odelberg, E., The behavior and qualities of the electric pig iron in the open-hearth process: *Metall. Chem. Eng.*, vol. 9, 1911, p. 509.

the electric furnace is less suited to the production of open-hearth steel than is normal blast-furnace pig iron.

It must not be inferred from the above statements that it is not possible to produce normal pig iron in the electric furnace. As a matter of fact some hundreds of tons have been so produced in this country and in Sweden. Although the furnace at Trollhättan is so operated as to produce "pig steel," because this metal is better suited to steel making than is normal pig iron, the Noble Electric Steel Co., of California, on the other hand, so operates its furnaces as to produce a soft foundry iron which is particularly suited to the foundries of that section of the country. However, it has been found that a greater output may be obtained when making "pig steel" per unit of electrical energy expended than when foundry iron is produced.

AN ESTIMATE FOR THE ERECTION OF AN ELECTRIC IRON-SMELTING AND STEEL-REFINING PLANT.

An estimate of the equipment and cost of erection of an electric iron-smelting and steel-refining plant, with estimates of the cost of production of pig iron and steel in such a plant is here presented from data furnished by Electro-Metals, of London.

ASSUMPTIONS AND CONDITIONS.

The estimate is based on the following assumptions:

OUTPUT.

Annual production of pig iron, 50,000 tons; annual production of steel ingots, 50,000 tons.

SIZE AND NUMBER OF FURNACES.

Five electric shaft furnaces of 3,500 electric horsepower.

One tilting, open-hearth furnace of 50 tons capacity heated by gas from the shaft furnaces.

Three electric steel furnaces of 1,500 electric horsepower and of 10 tons capacity, one being kept as reserve.

ELECTRIC POWER.

For electric shaft furnaces: 18,000 electric horsepower (includes 500 electric horsepower for gas blowers, motors, etc.).

For electric steel furnaces: 3,000 electric horsepower.

The price of electric power is assumed to be \$12 per electric horsepower-year. The electric power, 3-phase, of preferably 25 periods, is to be delivered at a high voltage from the water-power plant and to be transformed to a voltage of 50 to 100 volts close to the furnaces. For distribution to the electric motors a higher voltage of 500 to 1,000 volts should be used.

ORE.

To be supplied from local mines.
 Iron content, about 60 per cent.
 Cost, \$0.50 per ton at the works.
 Total quantity smelted per year, 80,000 tons.

FUEL.

For electric shaft furnaces, either coke or charcoal from local distilleries of wood alcohol.

Cost of fuel, \$5 per ton at the works.
 Total quantity consumed per year, 17,000 tons.

LIMESTONE.

From local quarries, lime rock, CaCO_3 content, 96 per cent.
 Total quantity used per year, 8,000 tons.
 Cost, \$1.50 per ton, including calcining.

PROPOSED SYSTEM OF WORKING.

The pig iron produced in the electric shaft furnace is transferred either by ladle or, better, direct through launders, into a tilting open-hearth furnace, in which by addition of ore and lime the greater part of the carbon is removed and the temperature raised to about $1,500^\circ\text{C}$. The open-hearth furnace has a capacity of 50 tons, and also serves as a mixer, from which at regular intervals about 10 tons of partly refined steel is transferred to the electric steel furnaces. In these the metal is further heated to $1,600^\circ$ to $1,700^\circ\text{C}$. and refined from any excess of sulphur not removed in the open-hearth furnace. The chief advantage of the electric steel furnace for the final treatment lies in the possibility of maintaining a reducing atmosphere in which reduction of the steel is more complete than in any other kind of steel furnace.

After being treated in the electric steel furnace for 1 to 2 hours the metal is recarburized to the desired carbon content, the ferro-alloys are added, and the metal is cast in the molds. The estimate provides also for fully equipped repair shops.

ITEMS ENTERING INTO ESTIMATED COST OF CONSTRUCTION.

In estimating the cost of construction of the plant the following items have to be taken into consideration:

Excavations for buildings and furnaces, grading and fencing, railway connection to works with crane for handling materials.

Light railway inside works with turntables, carriages, weigh-bridge, etc.

Water-supply plant with overhead storage tank and water pipes to gas scrubber, furnace water jackets and transformers, and pump with electric motor.

Buildings for furnaces and transformers.

Hoists for ore and lime, storage building for coke, crusher house and hoist.

General stores, offices, and laboratory.

Houses for officials and workmen.

Five electric shaft furnaces for 3,500 estimated horsepower, with foundations, iron constructions, brickwork, and lining.

Electrode holders with water jackets, lifting and regulating arrangements for the electrodes.

Gas blowers with electric motors, gas scrubbers, gas piping.

One tilting open-hearth furnace of 50 tons capacity with rockers and stands, hydraulic tilting cylinder, port ends, chills, gas and air connections and valves.

Working staging.

Iron chimney.

All silica, magnesite, fire, and red brick, silica and fire clay for brick setting, and magnesite.

Three electric steel furnaces, each of 10 tons capacity and 1,500 estimated horsepower, with foundations, iron construction, brickwork and lining, electrode holders with water jackets, automatic regulators, hydraulic tilting cylinders, working stagings, etc.

Two electric cranes of 20 tons carrying capacity.

Transformers for electric shaft and steel furnaces, with oil coolers, regulators, instruments and energy meters, lighting arresters, cables between transformers and furnaces.

Crushing plant with ore and coke crushers and electric motors.

Rails, wagons, weighing machines, tools, and sundries.

Transformers for small motors and for light, including wiring accessories.

Laboratory outfit and office fittings.

Superintendence during construction, drawings, and license.

Contingencies.

The total cost of erection of a plant of this description is estimated at \$850,000.

COST OF PRODUCTION.

The estimated cost of producing pig iron and steel in a plant of this description is presented below :

Estimated cost of producing 50,000 tons of electric pig iron a year.

Item.	Weight.		Cost.		
	Per ton. of ore.	Total.	Per unit.	Total.	Per ton of pig iron.
Ore, tons.....	1.6	80,000	\$1.50	\$120,000	\$2.39
Coke, tons.....	.33	17,000	5.00	85,000	1.68
Lime, tons.....	.16	8,000	1.50	12,000	.20
Electrodes, pounds.....	15	335	75.00	25,125	.52
Power (18,000 horsepower year).....			7.50	135,000	2.72
Staff and labor:					
3 foremen, 2 electricians, and 105 workmen.....				75,000	1.50
Maintenance and repairs.....				15,625	.31
Laboratory.....				3,125	.06
Office expenses.....				3,125	.06
Depreciation.....				28,125	.56
Royalty.....				37,500	.75
Contingencies.....				25,000	.50
Total.....				564,625	11.25

Estimated cost of producing 50,000 tons of electric steel ingots a year.

Item.	Weight.		Cost.		
	Per ton of steel.	Total.	Per unit.	Total.	Per ton of steel.
Pig iron, tons.....	1,000	50,000	\$11.29	\$564,625	\$11.29
Iron ore, tons.....	0.112	5,600	1.50	8,400	.16
Burnt lime, tons.....	.1	5,000	3.00	15,000	.29
Ferro-alloys, tons.....				25,000	.50
Electrodes, pounds.....	20	450	75.00	33,750	.68
Power (3,000 horsepower year).....			7.50	22,500	.45
Staff and labor:					
3 foremen, 2 electricians, 50 workmen.....				37,500	.75
Maintenance and repairs.....				37,500	.75
Ladles, stoppers, molds, and other tools.....				50,000	1.00
Laboratory and office expenses.....				6,250	.12
Depreciation.....				28,125	.56
Royalty.....				18,750	.37
Contingencies.....				31,250	.62
Total.....				878,650	17.54

THE ELECTRIC FURNACE IN STEEL MANUFACTURE.

By ROBERT M. KEENEY.

INTRODUCTION.

The purpose of the second part of this report is to present a brief historical review of the development of the electric-furnace manufacture of steel up to the present time; to describe in detail the types of electric furnaces in commercial operation for the manufacture of steel and, in general, types which have not yet attained wide use; to give a description of the practice of European and American electric-furnace steel plants that were recently visited by the writer; to compare in a general way the different types of furnaces and the more established methods of steel manufacture with the electric-furnace process; and to discuss various problems of the electric-furnace manufacture of steel and the possible future developments of the process.

HISTORY OF THE DEVELOPMENT OF THE ELECTRIC STEEL FURNACE.

EARLY DEVELOPMENT OF THE ELECTRIC FURNACE.

The process of evolution through which the electric furnace passed before reaching its most highly developed use, which is the manufacture of steel, really began with the furnace of Siemens, the prototype of the modern arc furnace. In 1878 Siemens patented the furnace and operated it in a very small laboratory way. This electric furnace consisted of a crucible of refractory material with a movable vertical electrode, the other electrode being an iron bar passing through the bottom of the crucible. One electrode was connected to the positive pole and the other to the negative pole of a direct-current circuit. The crucible was surrounded by a heat insulator, and the bottom metallic electrode was water cooled.

In 1885 Ferranti devised a furnace the principles of operation of which were the fundamental basis of the modern induction furnace. He used the induced current of a magnetic circuit to heat and melt metals which were arranged like the secondary of a short-circuited transformer.

Although the electric arc and induced currents were applied to furnace heating by these two pioneers, no attempt was made to operate a commercial electric furnace for the manufacture of steel until 1898. In the meantime, in 1886, the Cowles brothers had devised their arc furnace for the production of aluminum alloys. In 1887 Colby patented an induction furnace for melting metals that was quite similar to the modern Kjellin furnace. In 1892 Willson designed a furnace for the manufacture of calcium carbide that was simply a Siemens crucible reproduced on a commercial scale. In 1893 the radiated heat of the electric arc from two horizontal electrodes was used by Moissan during the course of his extensive experiments on carbides and oxides of metals. From 1893 to 1898 the electric arc furnace was developed rapidly in the manufacture of calcium carbide, although it was still essentially the old Siemens crucible with an upper electrode of carbon and a carbon block in place of the lower metallic electrode.

In 1898, owing to overproduction of calcium carbide and the substantiation of the Bullier patents, many works were compelled to stop the manufacture of carbide, and either turn to the manufacture of other products or shut down, which meant the idleness of many hydroelectric plants capable of developing thousands of horsepower. The experiments of Moissan had shown the possibility of making ferro-alloys in the electric furnace. Ferro-alloys were made in the old carbide furnaces, and with the introduction of these electric-furnace alloys, which were of higher percentage and of greater purity than the blast-furnace product, a steady demand for them arose. As a result, by 1900 a new industry was in operation. In many plants both calcium carbide and ferro-alloys were made, as is the case as present.

EARLY DEVELOPMENT OF THE ELECTRIC STEEL FURNACE.

DEVELOPMENT OF THE HÉROULT STEEL FURNACE.

As the manufacture of ferro-alloys developed and the necessity of producing alloys of low carbon content became apparent the design of the electric furnace was gradually altered so as to make the operation possible without a conducting carbon lining. It was only a step from the Héroult ferrochrome furnace lined with chromite, with a bottom carbon electrode, over which metal was permitted to freeze to prevent carburization, to the nonconducting hearth of the modern Héroult steel furnace, shown in figure 18, as it had been found that this freezing of metal hindered the regular operation of the furnace. The fundamental principles of the modern Héroult steel furnace were patented in 1900.

EXPERIMENTS OF STASSANO.

In the meantime Stassano,^a of Italy, through an interest in the carbide industry, had attempted to produce steel directly from iron ore in the electric furnace, with charcoal as a reducing agent. The experiments were begun in 1898 and were first conducted in a modified blast furnace, in which electrodes were substituted for tuyères. This furnace being unsuccessful, a new furnace similar to the modern Stassano steel furnace, shown in figures 19 and 20, was built, having horizontal electrodes and arcs heating the charge by radiation, as in

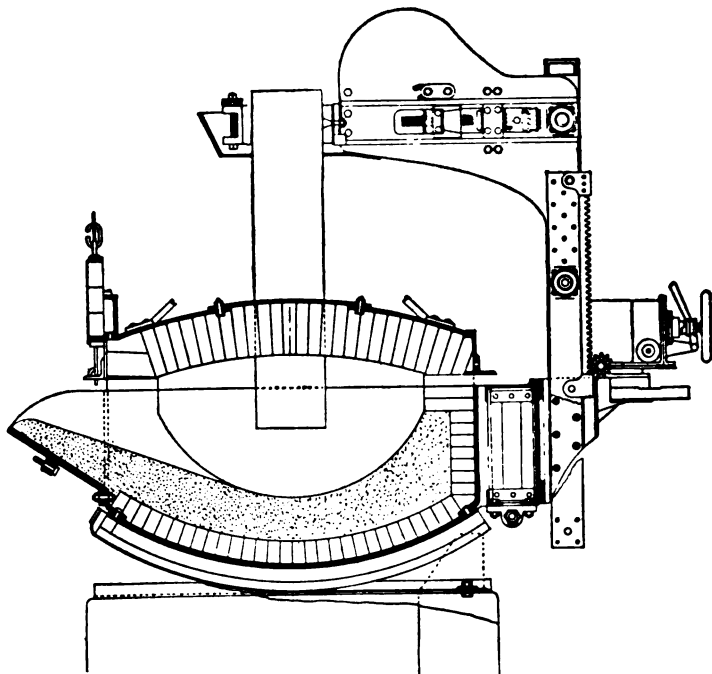


FIGURE 18.—Sectional elevation of 2.5-ton, single-phase Héroult steel furnace, La Praz, France.

the Moissan furnace. However, it was not well suited to reduction of the ores that were first used.

In these experiments a high-grade hematite ore of low phosphorus and sulphur content was used. The ore was first crushed and then briquetted with charcoal, lime, and tar. The steels produced contained about 0.10 per cent carbon, less than 0.024 per cent phosphorus, and not over 0.073 per cent sulphur. The power consumption aver-

^a Stassano, E., Treatment of iron and steel in the electric furnace: *Electrochem. and Metall. Ind.*, vol. 6, 1908, p. 315; Catani, R., The application of electricity in the metallurgical industry of Italy: *Jour. Iron and Steel Inst.*, vol. 84, No. 2, 1911, p. 215; *Metall. Chem. Eng.*, vol. 9, 1911, p. 642.

aged 4,205 kilowatt-hours per metric ton.^a It will be seen later that this item of power consumption is over five times that of making steel from cold scrap in the electric furnace, a difficult handicap for

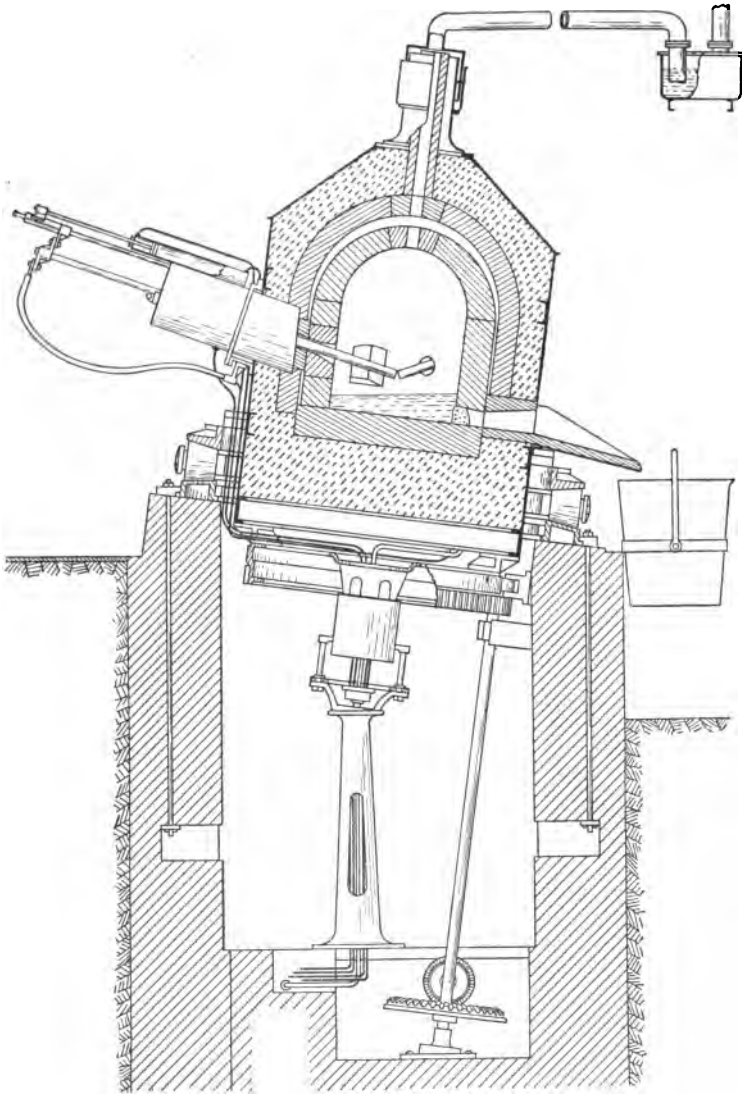


FIGURE 19.—Elevation of 1-ton, three-phase Stassano steel furnace, Bonn, Germany.

any direct electric-furnace steel process to overcome, except under the most favorable circumstances. Later experiments of others have reduced this item considerably.

^a Throughout this report the metric ton, 2,204 pounds, is used in reference to European practice, and the long ton, 2,240 pounds, for American practice.

Stassano was the first to demonstrate that malleable iron or a fluid steel can be produced directly from a pure ore in the electric furnace. The power consumption was so high and the cost of briquetting so great as to prevent the commercial application of this process. Later, in 1908, Stassano showed by experiments the possibility of obtaining good steel from ore containing impurities, but the cost was still prohibitive.

From these experiments on iron ore, Stassano turned to the development of his furnace for the manufacture of steel from scrap iron and steel, and he had a furnace operating for this purpose at the Italian Government gun foundry in Turin by the time of the visit of the Canadian Commission in 1904.

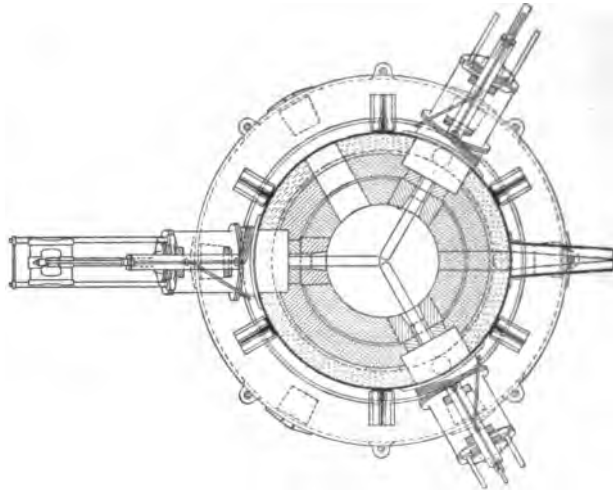


FIGURE 20.—Plan of 1-ton, three-phase Stassano steel furnace, Bonn, Germany.

THE KJELLIN FURNACE.

The application of the induction principle to the manufacture of steel was begun not because of an overdevelopment of any electric-furnace industry, but because of actual need of such a furnace in steel manufacture. In 1899 Kjellin, a Swedish electrical engineer, was requested by the works manager at Gysinge, Sweden, to construct an electric furnace. The furnace was to be lined for the production of the highest grade of tool steel from Dannemora iron manufactured at Gysinge. Kjellin did not approve of the arc electric furnace used in carbide manufacture, because he thought the contact of the electrodes with the charge would contaminate the product. A furnace was designed in which the molten steel lay in an annular ring that was used as the secondary coil of a step-down transformer. Kjellin conceived the construction independently and

did not know until later, when he applied for patents, that the same principle had previously been used for this purpose by Ferranti and Colby. To Kjellin, however, belongs the credit of constructing the first induction steel furnace that was a commercial success. Construction was started in May, 1899, and completed in February, 1900. On March 18, 1900, the first steel ingot was cast. The small 58-kilowatt furnace of 100 kilograms (220 pounds) capacity produced about 600 kilograms (1,320 pounds) of ingots per 24 hours, and was successful from the start. A larger furnace taking 165 kilowatts and producing 4 tons per 24 hours was started in May, 1902. In fundamental design these furnaces were practically the same as the modern Kjellin electric steel furnace (fig. 21).

INVESTIGATIONS OF THE CANADIAN COM- MISSION OF 1904.

In 1904 the commission^a appointed by the Canadian Government to investigate electric-furnace pig iron and steel processes in Europe found four electric furnaces being used for the production of steel at the following places: A Kjellin furnace at Gysinge, Sweden; a Héroult furnace at Kortfors, Sweden; a Héroult furnace at La Praz, France; and a Stassano furnace at Turin, Italy.^a Such was the nucleus from which the modern manufacture of steel in the electric furnace has developed.

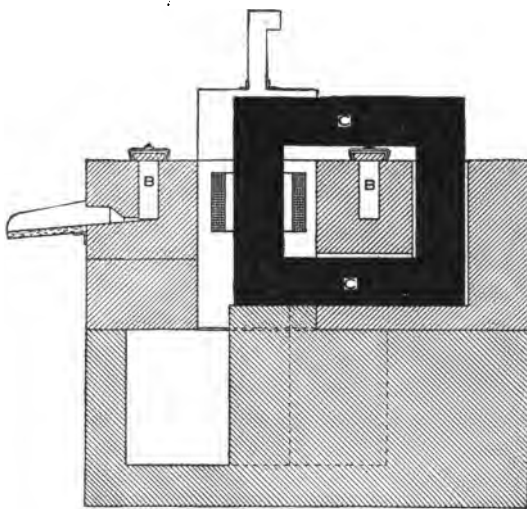


FIG. 21.—Elevation of 1.5-ton, single-phase Kjellin steel furnace, Gysinge, Sweden. C, laminated iron core of primary circuit; B, annular crucible.

EXPERIMENTS WITH KJELLIN FURNACE, GYSINGE, SWEDEN.

At Gysinge three experiments were conducted in a 165-kilowatt Kjellin furnace of the type shown in figure 21. The first charge was arranged to give a high-carbon steel containing about 1 per cent of carbon. The charge consisted of high-grade Dannemora pig iron, bar scrap Walloon iron, and some scrap steel left in the furnace from previous runs. All of the materials had a low phosphorus

^a Haanel, E., Report of the commission appointed to investigate the different electro-thermic processes for the smelting of iron ores and the making of steel in operation in Europe: Mines Branch, Department of the Interior, Canada, 1904, p. 1.

and sulphur content. After 6 hours' running about 1 ton of very high-grade steel containing 1.082 per cent carbon was poured. There was very little slag removed in the process. The power consumption was 857 kilowatt-hours per ton.

The object of the second experiment was to make a medium carbon steel containing about 0.5 per cent carbon. The same grade of material was charged as in the first run but in different proportions. After 6½ hours' running about 1 ton of very pure steel, containing 0.417 per cent carbon, was poured. A little more slag was removed than in the first run. The power consumption was 1,100 kilowatt-hours per ton.

In the third experiment the object was to make a low-carbon steel containing less than 0.2 per cent carbon. The charge consisted of bar iron with a low carbon content. Three attempts were made to get a clean pour, but owing to the power on the furnace being insufficient to give a temperature high enough for this grade of steel no precise results were obtained. The steel poured contained 0.098 per cent carbon and had a low phosphorus and sulphur content. A little less than 1 ton of metal was obtained with a power consumption of 1,430 kilowatt-hours per ton.

Kjellin estimated the power cost at this plant as \$15.50 per kilowatt-year, or 0.18 cent per kilowatt-hour. On this basis the cost of production of a high-carbon steel of the first class was estimated at \$37.48 per ton, or \$5.82 per ton exclusive of the cost of material.

The process as conducted in 1904 was, as regards its metallurgical features, similar to the old crucible process, the desired variation in the finished steel being produced by altering the relative proportions of pig iron and scrap iron and little or no purification being attempted. The commission found the Kjellin furnace well adapted for the production of the highest class of steel from pure materials. It seemed to be limited in use to pure materials, and gave better results with high-carbon steel than with mild steel. There was a marked increase in power consumption when the percentage of carbon in the steel was decreased.

EXPERIMENTS WITH THE HÉROULT FURNACE, LA PRAZ, FRANCE.

At La Praz, France, two experiments were made on the production of both high and low carbon steel in a 320-kilowatt Héroult single-phase furnace with a capacity of 2½ to 3 tons (fig. 18), described on page 74. In the first run the materials consisted of scrap iron and steel containing 0.11 per cent carbon, 0.055 per cent sulphur, and 0.22 per cent phosphorus, and a slag of iron ore and lime for dephosphorizing. When completely melted the slag was poured off and two successive slags of lime, sand, and fluorspar were added to remove the sulphur. The charge was poured after a run of 4½ hours. About 1½ tons of excellent steel was obtained, containing 0.079 per cent

carbon, 0.009 per cent phosphorus, and 0.022 per cent sulphur. This steel was for transformers. The yield was 84 per cent of the total charge of metal. The power consumption was 1,555 kilowatt-hours per ton.

The object of the second experiment was to produce a high-carbon steel. The charge was about the same as in the first, but after complete purification at the end of 5 hours and 20 minutes the bath was recarburized by the addition of carburite, a mixture of pure carbon and iron. Eight hours after starting about 2½ tons of steel was poured. This yield was about 90 per cent of the charge of metal. The steel contained 1.016 per cent carbon, 0.02 per cent sulphur, and 0.009 per cent phosphorus. The power consumption was 2,840 kilowatt-hours per ton.

The electrode consumption was about 18 kg. (39.6 pounds) per ton. There was a 50 per cent loss in electrodes due to unconsumed ends. These, however, were mixed with an equal quantity of fresh coke to make new electrodes. Harbord^a estimated the cost of converting scrap into steel at La Praz at \$15.40 per ton of product exclusive of materials.

The operation at La Praz was decidedly different from that at Gysinge; at Gysinge high-grade steel was produced from pure materials, while at La Praz a high-class steel was made from low-grade scrap. The high power consumption at La Praz was due to the necessarily prolonged use of dephosphorizing slags. When scrap iron nearly free from carbon was used, the first product obtained was soft steel; to obtain high-carbon steel suitable additions of carbon had to be made. For this reason the consumption of energy was greater with high-carbon than with low-carbon products. The process at La Praz was the reverse of the one at Gysinge, where a high-carbon material was available at the start. The methods depend on the materials available. Although fine raw material had to be used at Gysinge, any sort of scrap could be used in the Héroult furnace because of the high temperature and the easy manipulation of slags. The commission found the Héroult furnace admirably suited to the manufacture of high-grade tool steel from impure materials.

CONCLUSIONS OF THE COMMISSION.

The conclusions reached in 1904 by Harbord,^b metallurgist of the commission, in respect to the electric production of steel, were as follows:

1. Steel equal in all respects to the best Sheffield crucible steel can be produced either by the Kjellin or Héroult processes at a cost considerably less than the cost of producing high-class crucible steel.

^a Harbord, F. W., Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in operation in Europe; Mines Branch, Department of the Interior, Canada, 1904, p. 50.

^b Harbord, F. W., loc. cit., p. 115.

2. Structural steel to compete with Siemens or Bessemer steel can not be economically produced in electric furnaces, and such furnaces can be used commercially only for the production of exceptionally high-class steel for special purposes.

LATER DEVELOPMENT OF THE ELECTRIC STEEL FURNACE.

THE GIROD ARC FURNACE.

In connection with the development of the Girod ferro-alloys works at Ugine, France, Girod had devised an arc furnace with a noncarbon-

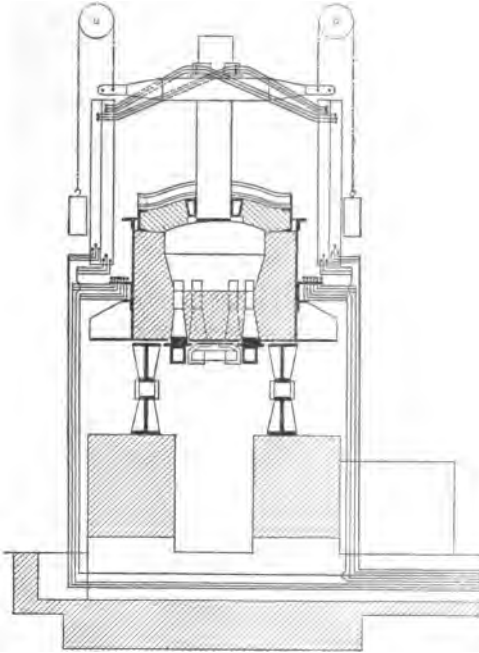


FIGURE 22.—Elevation of 2.5-ton, single-phase Girod steel furnace showing arrangement of conductors.

conducting hearth that was used for the manufacture of low-carbon alloys. As the ferro-alloy industry became well established, Girod turned his attention to a steel furnace, and in 1905 constructed a furnace based upon the principles of the earlier ferro-alloy furnace. The furnace (fig. 22) has a conducting hearth consisting of magnesite, in which iron poles are embedded to serve as conductors. The poles are water-cooled at the lower ends. There may be one or more upper carbon electrodes. Thus the chief feature of the Girod furnace may be traced back to the lower water-cooled iron electrode of Siemens.

Girod was the first to apply the noncarbon conducting hearth to the steel furnace.

OTHER ARC FURNACES.

After the introduction of the noncarbon conducting hearth in 1905 other furnaces similar to the Girod were developed. The Keller furnace (fig. 23) is similar to the Girod furnace, except that there are numerous iron rods embedded in the hearth of magnesite about $1\frac{1}{4}$ inches apart, instead of about six poles, as in the Girod furnace. The whole mass of magnesite and iron acts as a conductor when hot.

Grönwall constructed a furnace (figs. 24, 25) having a conducting hearth composed of a mixture of dolomite and tar, and two upper carbon electrodes which were each connected to a phase of a two-phase system, the hearth forming the neutral point of the system.

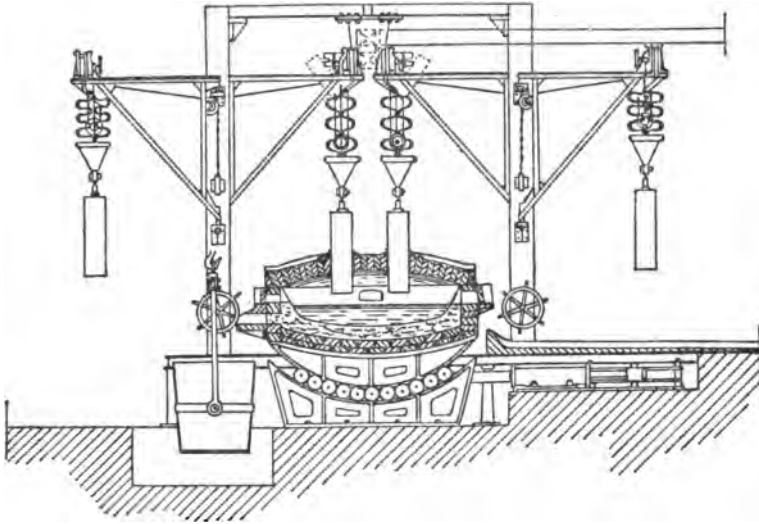


FIGURE 23.—Elevation of 8-ton, three-phase Keller steel furnace, Unieux, France.

bon electrodes which were each connected to a phase of a two-phase system, the hearth forming the neutral point of the system.

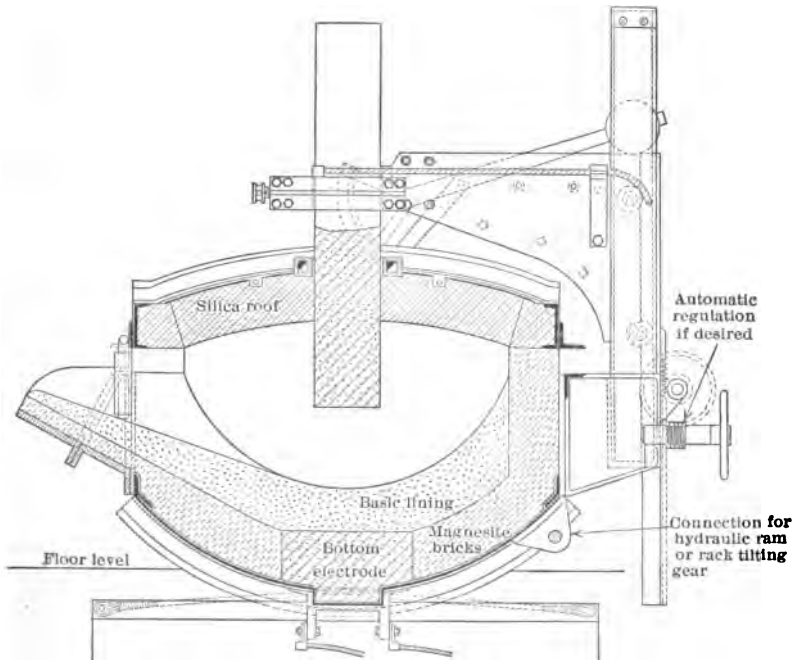


FIGURE 24.—Elevation of 2.5-ton, two-phase Grönwall steel furnace, Sheffield, England.

Nathusius has built a furnace (fig. 26) combining the principles of the arc and resistance furnaces, as the hearth contains three poles

of iron, each connected to a phase of a three-phase system, while there are also three upper carbon electrodes, each connected to a phase. This is the latest development of the arc-resistance furnace.

Other furnaces of the arc type which have been designed and built since 1904 are the Anderson, Chaplet, and Soderburg furnaces.

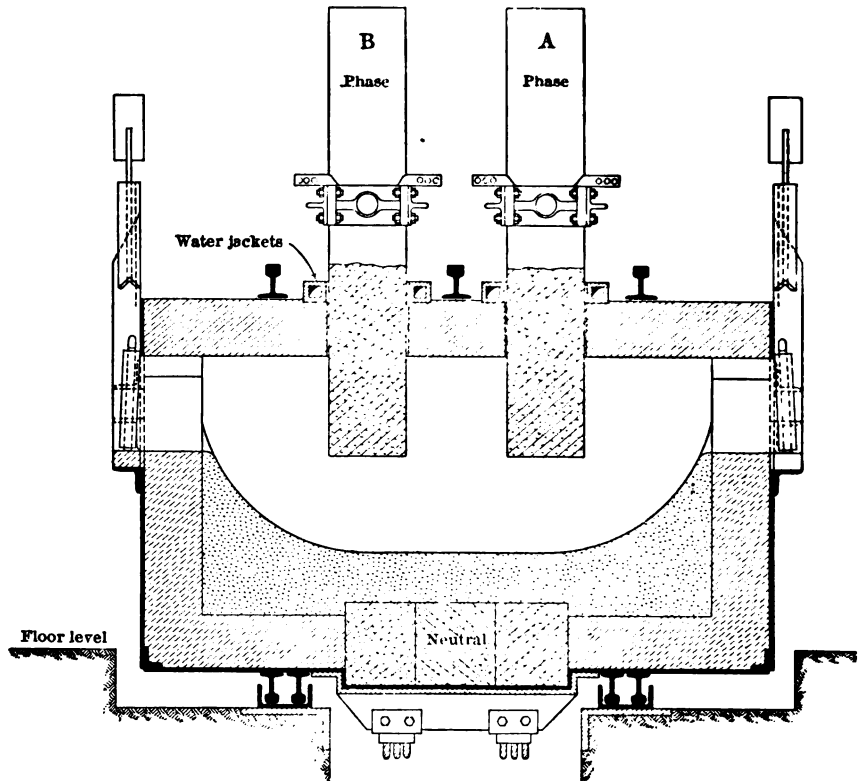


FIGURE 25.—Longitudinal elevation of 2.5-ton, two-phase Grünwall steel furnace, Sheffield, England.

THE RÖCHLING-RODENHAUSER FURNACE.

The Röchling-Rodenhauser electric steel furnace (fig. 27) was developed from the original Kjellin design in 1906 to meet the requirements for refining molten basic Bessemer steel. It combines two methods of electric heating—the ordinary heating by electric induction currents and heating by induction currents that are introduced into the molten bath through metal plates. It is the most widely adopted induction furnace of the present time, and the design is practically the same as the original. The Röchling-Rodenhauser

furnace was the first induction furnace that could be heated with a current of ordinary frequency without having too low a power factor, as had the old design.

OTHER INDUCTION FURNACES.

Other induction furnaces are the Frick, the Hiorth, and the Paragon furnace of Harden. Several furnaces of the first two types have been built, but their adoption has not been widespread. Greene^a has devised a combination induction converter which has not passed beyond the experimental stage.

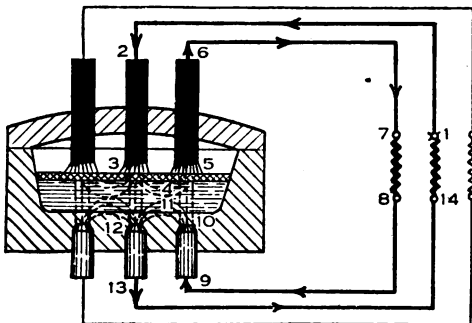


FIGURE 26.—Elevation of Nathusius three-phase steel furnace showing principle of operation.

GENERAL LINES OF DEVELOPMENT OF ELECTRIC STEEL FURNACES.

With the natural increase in size of units constructed there has been a marked tendency to use three-phase electric current in the

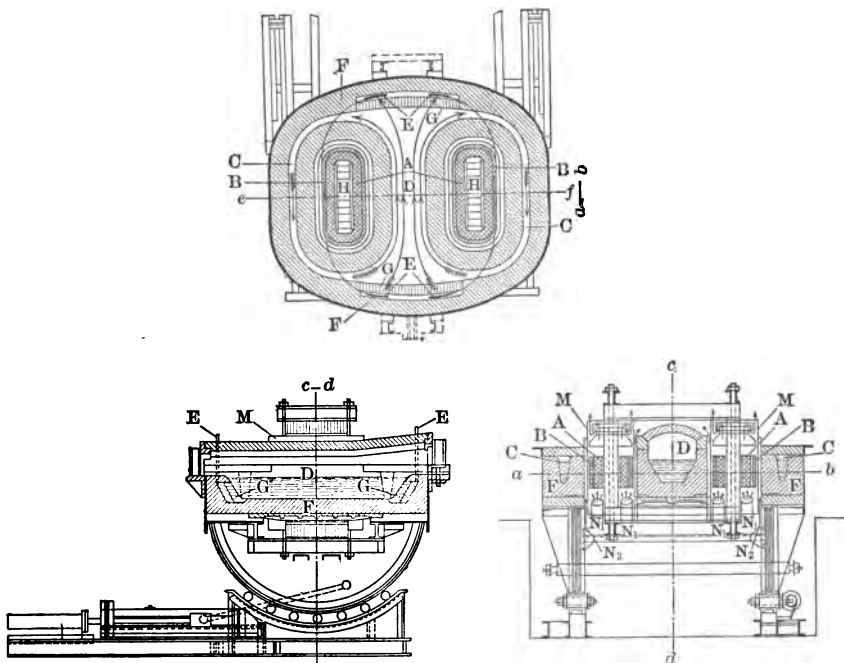


FIGURE 27.—Plan and elevation of 2-ton, single-phase, Röchling-Rodenhauser steel furnace, Völklingen, Germany. For explanation of lettering see p. 86.

larger furnaces of all types, except the induction furnace. Many of the recent furnaces have been designed especially with this in view

^a Greene, A. E., Electric steel processes as competitors of the Bessemer and open-hearth; *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 233.

to save the expense of installing motor generator sets for transformation from three or two phase current to single-phase current. At present new generating units are invariably three-phase if the energy is to be transmitted a considerable distance. It is probable that a more even distribution of heat is obtained in a large furnace by the use of three-phase current. In induction furnaces it has been proved by experience that for units above 3 tons three-phase current does not give as good satisfaction as single-phase current, owing to the complicated nature of the electrical transforming part of the furnace.

Electrode consumption has been reduced considerably as a result of the production abroad of amorphous carbon electrodes in sizes up to 24 inches (61 cm.) in diameter, that screw into each other so that the butts are not wasted. Graphite electrodes of this design have been made in this country, but up to a recent date no domestic amorphous carbon electrodes threaded for continuous feeding were made. The strength and density of the carbon electrode are considerably greater than they were several years ago. The carbon electrode is used almost entirely by foreign electric-steel manufacturers, because of its greater cheapness.

Generally speaking electric steel furnaces were first operated by men not familiar with the steel industry, so that linings and roofs were not always properly constructed. The rather high cost item that resulted has been reduced considerably with the adoption of the electric furnace by established steel firms.

Under the hand of the steel manufacturer, the electric furnace seems to be gradually taking the form of the open-hearth furnace. The hearth proper of the Héroult furnace is like an open hearth, but the furnace is charged from the sides and front and is generally set back a short distance from the edge of the working floor. In the open-hearth furnace the charging door is on one side and the pouring spout on the opposite side, the furnace being level with the edge of the working floor. The electrode supports of the Héroult furnace as at present arranged prevent the use of a back charging door. The Girod furnaces at Ugine, however, are similar to the open-hearth furnace in these respects, as the electrodes are supported from the sides. A three-phase Héroult furnace now being constructed in England is to have the three electrodes suspended from the sides as in the Girod furnace, with a back charging door opposite the spout and close to the edge of the working floor. The Kjellin furnace is simply an annular crucible. The Röchling-Rodenhauser furnace has a rather large hearth in the center, and a charging door opposite the pouring spout.

The wider use of the electric process, together with the improvements mentioned, has resulted in a considerable increase in heating efficiency over the results obtained in 1904.

For a few years the induction furnace was developed as rapidly as the arc furnace, but during the past three years its use has not increased as rapidly. The reason for this will be discussed later.

PRESENT STATUS OF THE ELECTRIC STEEL INDUSTRY.

In 1904 there were 4 electric furnaces being used in Europe for the manufacture of steel, whereas to-day there are 114 electric furnaces producing steel in Europe and the United States and 30 others are in course of construction. As in other electrothermic processes, development has not been so rapid in the United States as in Europe. Only 14 furnaces are in this country. The average capacity per charge of the furnaces already built is 3.7 tons, whereas that of the furnaces under construction is 4.5 tons, an increase of 21.6 per cent. The total charge capacity of the furnaces now installed is about 250 tons per charge, and the total charge capacity of the furnaces under construction will be 170 tons per charge. The arc furnaces vary in capacity from 1 to 15 tons and require from 200 to 1,500 kw. for operation. A Héroult furnace of 25-ton capacity, requiring 3,000 kw., is nearly completed at Bruckhausen, Germany. The induction furnaces vary in capacity from 1 to 10 tons and require from 165 to 600 kw. for operation.

Of the 114 furnaces in operation 84 are arc furnaces and 30 are induction furnaces; of the 30 under construction, 26 are arc furnaces and 4 induction furnaces.

The following table gives the annual production of steel in electric furnaces, by countries, for the years 1908 to the first half of 1912.

TABLE 1.—Yearly production of electric-furnace steel.

Country.	1908			1909			1910			1911			1912, first half.
	Produc- tion.	Number of fur- naces.	Change in pro- duction.	Produc- tion.	Number of fur- naces.	Change in pro- duction.	Produc- tion.	Number of fur- naces.	Change in pro- duction.	Produc- tion.	Number of fur- naces.	Change in pro- duction.	
	Tons.		Per cent.	Tons.		Per cent.	Tons.		Per cent.	Tons.		Per cent.	Tons.
Germany and Luxemburg.....	19,586	8		17,773	8	— 9.0	36,188	13	+104.1	60,654	15	+57.8	
United States.....	85	1		13,762	4	+109.2	52,131	7	+328.9	28,165	9	— 44.2	6,883
Austria-Hungary.....	4,333			9,048		+109.2	20,028		+120.9	23,897	10	+3.0	
France.....	2,686	7		6,515	12	+127.6	15,449	21	+104.2	13,850	21	+3.0	7,920
Sweden.....				591	11		35,431	12	— 27.0	2,034	13	+372.0	
Norway.....								1					
Denmark.....													
England.....											12		
Italy.....					5			2			4		
Switzerland.....													
Belgium.....											2		
Russia.....											1		
Total.....	26,610	16		47,039	40	+78.2	120,116	56	+155.4	128,476	90	5.22	

From Table 1 it may be seen that for a new process in so conservative an industry as iron and steel manufacture, progress has been very rapid since 1908. Germany leads all countries in the steady growth of the process and the total tonnage produced. Although in Germany the production of electric-furnace steel increased 67.8 per cent in 1911, in the United States it decreased 44.2 per cent. The decrease in this country was probably due to the conservatism of American steel makers, which has prevented the wide adoption of the process before experimental results have conclusively proved its merits. From present indications there will be a considerable increase in the production of electric-furnace steel in this country in the near future, although a very small tonnage, 6,882 tons, was reported by the American Iron and Steel Institute as the output for the first half of 1912.^a Of the total production of electric-furnace steel in the United States in 1911, 27,227 tons were ingots and 1,878 tons castings. Of the total tonnage of electric-furnace steel made here in 1911, 6,700 tons were alloy steel and 462 tons were rolled into rails. The large production of steel in the United States and Germany in proportion to the number of furnaces operating is due to the use of molten Bessemer and open-hearth steel instead of cold scrap. The use of the latter almost entirely accounts for the comparatively small tonnage produced by France in proportion to the number of furnaces in operation. No figures were obtained for England, but it is probable that at least 10,000 tons of electric-furnace steel is manufactured in England. It is estimated that about 12 furnaces operate there, several of which receive hot-metal charges. Italy, Norway, Switzerland, Belgium, and Russia produce small tonnages also. The slight increase in the total electric-furnace steel production for 1911 over that produced in 1910 was caused by the big decrease in production in the United States.

In the first years of its development the electric process was considered as a competitor of the crucible process only for making high-class steel from scrap iron and scrap steel; but with the successful operation of larger furnaces the electric process is likely to become an important adjunct to the Bessemer and open-hearth processes as a means of superrefining the molten products that they yield. The electric process, however, does not appear to be destined to supersede either of these methods, as greater efficiency and economy are obtained by a combination of any two of the three processes as a duplex process. The success of recent experiments has obtained for the electric process a definite place as a superrefining method. In time preliminary refining will probably be done mainly in the Bessemer converter, the process being finished in the electric furnace or the open hearth. In Europe the electric-furnace process for making steel

^a Iron Age, The world's output of electric steel; vol. 91, 1913, p. 304.

of the highest grades is rapidly superseding the old crucible method, because of its greater economy of operation and the possibility of using materials of lower grade.

ELECTRIC STEEL FURNACES.

In general electric furnaces in commercial use for the manufacture of steel may be divided into two groups—arc furnaces and induction furnaces. In the arc furnace the heating is caused by the arc, which may be between the electrode and the bath, or between two or more electrodes so arranged as to heat the metal by radiation only. In the induction furnace the heat is supplied by a current induced in the bath. The operation is similar to that of a step-down transformer, having a large number of primary turns and a single secondary turn, which is formed by the steel in the furnace.

ARC FURNACES.

THE HÉROULT FURNACE.

The Héroult electric steel furnace heads the list of electric furnaces in use in the iron and steel industry with 31 furnaces already built and 20 others in course of construction. The wide use of the Héroult furnace is due chiefly to its efficiency, simplicity of construction, and adaptability to many different uses. Also it was the pioneer electric steel furnace.

SINGLE-PHASE HÉROULT FURNACE.

The design of the 2½-ton, single-phase Héroult furnace has changed little from that (fig. 18) of the first Héroult furnace, which has been in operation at La Praz, France, continuously since 1900. The furnace consists of a shallow hearth of dolomite, similar to the open hearth, incased in a steel shell, and covered with a roof of silica brick. The 2½-ton Héroult furnace at Braintree, England, has the pouring spout in the center of one side, and two doors, one on each end of the furnace. There are two 14-inch carbon electrodes projecting into the furnace through the roof and held in place by a steel framework extending from the rear over the roof. The electric current arcs between each of the electrodes, which are connected in series, and the bath, thus passing through the bath. The Braintree furnace is operated with 300-kw. 25-cycle alternating current at 100 volts. The furnace is set on curved steel bars, the whole being on a concrete foundation, and is tilted by rotating a screw, operated by a 5-kw. electric motor. Springs are used on the bottom to keep the furnace from creeping.

The exterior of the 2½-ton furnace is 5 feet 6 inches wide, 7 feet 6 inches long, 4 feet 9 inches high to the top of the roof, and 9 feet

3 inches high to the top of the electrode holders. The furnace foundation is 2 feet 6 inches above the main floor of the foundry. The whole furnace is incased in plate steel $\frac{3}{8}$ inch thick.

In lining the furnace a mixture of tar and dolomite is tamped in around a sectional mold to a depth of 9 inches to form the bottom. The sides of the hearth slope about 60 degrees up to about 18 inches above the bottom. From here up to the roof, 15 inches, the lining is magnesite brick. The lining is 12 inches thick on the sides and 14 inches thick on the ends. The internal dimensions of the furnace are 4 feet 2 inches by 1 foot 8 inches at the bottom of the hearth, and 5 feet by 2 feet 7 inches at the top, with a depth of 31 inches. The doors are 9 by 10 $\frac{1}{2}$ inches. The customary roof for this type of furnace is silica brick, set so as to give a roof 12 inches thick. At Braintree considerable difficulty has been experienced with the roof and several kinds of brick have been tried, such as silica, magnesite, and bauxite. Bauxite brick were being used recently.

The electrodes are threaded for continuous feeding and are used in either 4 or 6 foot sections. Each of the two electrode holders consists of two vertical 9-inch channel irons that are set 5 inches apart and act as guides for two $\frac{3}{8}$ -inch copper plates. These vertical copper plates are attached to a heavy horizontal copper plate that extends over the furnace and has a clamp tightened by a screw for supporting the electrodes. The electrodes may be raised or lowered by hand wheels at the rear or by automatic Thury regulators. In later furnaces the copper, that serves to support the electrode as well as conduct the current, is to be replaced by manganese steel supports and copper bus bars large enough to conduct the current. This will strengthen the holder and reduce the amount of copper, about 1 ton being necessary in the old design. The electrode holders are water cooled at the electrodes, and there are copper water jackets around the electrodes where they enter the roof.

The cost of a 2 $\frac{1}{2}$ -ton Héroult electric steel furnace may be estimated as follows:

The furnace, automatic regulators, platform, transformers, switch-board, and other equipment will cost when completely installed about \$12,000 to \$15,000, exclusive of crane or buildings; allow an engineering fee of \$5,000, and \$350 per month and expenses for an expert to establish the operation of the furnace, making a total of about \$25,000. The royalty on tool steel produced would be from \$1 to \$3 per ton according to quantity, and on castings, billets, or ingots 50 cents per ton.

THREE-PHASE HÉROULT FURNACE.

With the use of three-phase electric current in the Héroult furnace of 15 tons capacity a slightly modified design has been made. Until

recently the largest three-phase Héroult furnaces were the two in this country, one at South Chicago, Ill., and the other at Worcester, Mass., which are of identical design, but recently there has been erected a 25-ton three-phase Héroult furnace at Bruckhausen, Germany, and a 22-ton furnace is almost completed at the same plant.

The essential difference between the 15-ton, three-phase furnace at South Chicago, Ill.,^a and the single phase 2.5-ton Héroult furnace is that in place of two electrodes there are three let down through the roof so that their ends form the vertices of an equilateral triangle, each side of which is 5 feet 2 inches long, one vertex of this triangle pointing toward the back of the furnace. The center of this triangle coincides with the center of the furnace. Each electrode is connected to a phase of a three-phase circuit.

A steel overhead structure supports the electrodes, the weight being directly supported by chains which run back over pulleys on the framework to the drums at the back of the furnace. The electrodes are kept in alignment by vertical guides. The chains are attached to three separate solid copper holders, which are bolted directly to the bus bars. In front these holders are split and joined with a right-and-left screw, which enables the holder to be opened or closed at will. The holders can be made to carry any size of electrode up to 24 inches diameter by use of contact blocks.

The electrode may be regulated by three hand regulators about 4 feet back of the furnace, or by automatic regulators. There is an individual motor for each electrode and a complete automatic device similar to the Thury regulator.

The furnace proper has a shallow hearth, as in the single-phase furnace, but is circular instead of rectangular. The outside dimensions are approximately that of a complete circle 13.5 feet in diameter, with two flattened portions at the front and back. The furnace shell is of plate steel 1 inch thick, riveted together.

The furnace bottom is made of one row of magnesite brick laid on edge across the steel shell, over which is rammed dead-burned Spaeter magnesite to a depth of 12 inches at the center—its thinnest point. The side walls consist of two rows of magnesite bricks laid on end, giving a thickness of 18 inches up to the furnace roof. The roof is made of silica brick and is 12 inches thick. There is an 8-inch rise in the 10-foot span across the furnace.

The furnace has five doors, two on each side and one in front over the pouring spout, of cast iron lined with fire brick, 4½ inches thick. They are operated by steam pressure, with the exception of the one over the pouring spout, which is operated by hand with a counter-balance.

^a Osborne, G. C., The 15-ton Héroult furnace at the South Chicago works of the Illinois Steel Co.: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 205.

The foundation is of concrete and extends 5 feet above the ground. On the foundation is a stationary rack 8 feet 9 inches long, upon which the furnace proper rests on a floating pinion fastened to the shell by rivets. The arc of this floating pinion has a radius of 10 feet, which gives the furnace a tilting angle of 29° . Attached to the extreme back of the furnace is an 18-inch plunger with a 4-foot stroke working in a cylinder attached to a hydraulic line of 500 pounds pressure per square inch, which gives a lifting power of about 45 tons. The furnace rights itself by its own weight after tilting.

For operation the furnace takes 1,200 to 1,500 kw., supplied by a three-phase current at about 90 volts, and having a frequency of 25 cycles.

THE GIROD FURNACE.

The main difference between the Girod electric steel furnace and the Héroult furnace is that the former has a hearth which conducts the electric current. There are in operation 16 Girod furnaces of 2.5 to 12 tons capacity and 5 are in course of construction. This type of furnace seems to be especially satisfactory in the refining of cold scrap steel because of the slight fluctuations in power demand.

SINGLE-PHASE GIROD FURNACE.

The single-phase furnace at Ugine, France^a (fig. 28), has a shallow conducting hearth of dolomite with pieces of soft steel embedded in the dolomite near the periphery, and a carbon electrode passes through the roof. In the operation of the furnace current passes through the carbon electrode and through the steel bath, which touches the tops of the steel poles embedded in the hearth. The furnace is mounted on rollers and tilted by an electric motor. This furnace has a capacity of 2.5 to 3 tons and is operated with 300 kw. The voltage is 60 to 65 volts, and the frequency of the current is 25 cycles.

The hearth of the furnace is 3 feet square at the bottom and 6 feet square at the top. The roof is 31 inches above the hearth. The water-cooled steel electrodes (fig. 29), embedded in the hearth when new, project beyond the bottom of the hearth a short distance. There are 6 steel poles set on the circumference of a 31-inch circle. The floor of the hearth is made of dolomite and tar rammed in. The walls are magnesite brick; the roof is silica brick. The roof is insulated from the walls by a thin layer of asbestos. The whole hearth is encased in a $\frac{5}{8}$ -inch steel shell and is set on a concrete foundation that extends 6 feet above the ground level. The furnace has one

^a Borchers, W., Electric smelting with the Girod furnace: Trans. Am. Inst. Min. Eng., vol. 41, 1910, p. 120.

charging door at the rear and a pouring spout in front, and resembles the open-hearth furnace even more than the Héroult furnace does.

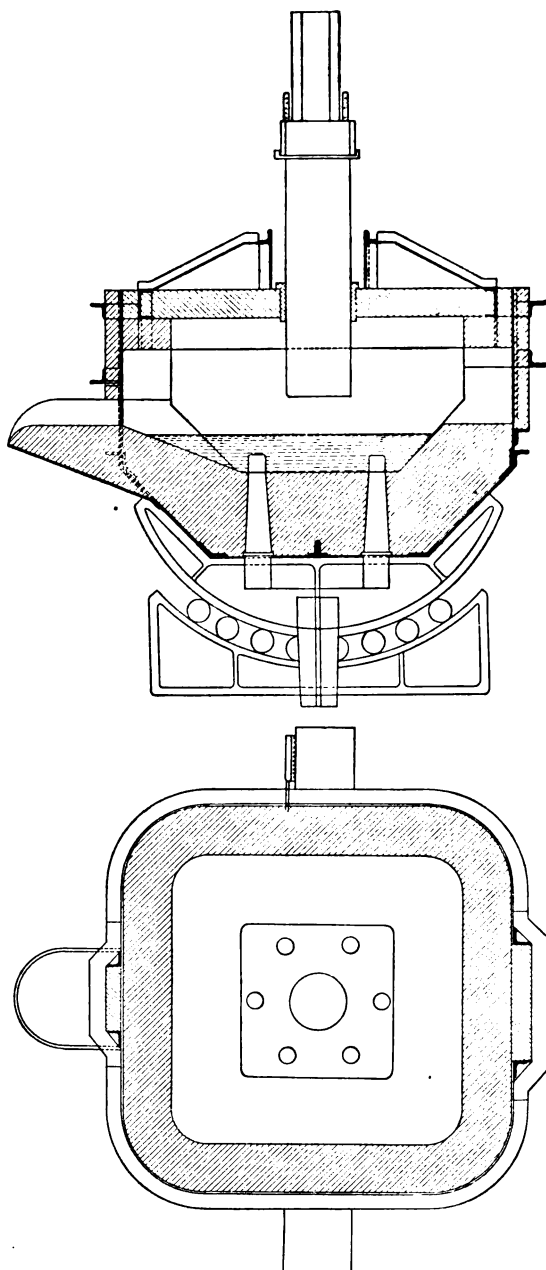


FIGURE 28.—Plan and elevation of 2.5 to 3 ton, single-phase Girod steel furnace, Ugine, France.

The 2.5-ton furnaces at Ugine are operated with carbon electrodes 14 inches in diameter and 5 feet long, some of which are threaded for continuous feeding. The means of supporting the electrode differs from the Héroult scheme in that the supporting steel structure is built over the furnace from the sides (fig. 22), making it possible to have a door at the rear. The crosspiece holds a water-cooled holder attached to the electrode. There is also a water jacket around the electrode where it passes through the roof. The crosspiece is raised or lowered by a screw at each end, the side structural work serving as a guide. The electrode may be adjusted by hand or by automatic control.

With a conducting-hearth furnace there is a considerable tendency to the presence of induced currents in the steel shell which reduce the power factor. Hence the arrangement of the electrical conductors is of great importance in the Girod furnace. Three meth-

ods have been used: (1) The shortest path from the motor-generator set to the carbon and steel electrode is used, so that all of the cables are on the side of the furnace that faces the motor generator. (2) The cable to the electrode is in two parallel sections, and the steel electrode is connected by the shortest path to the motor-generator set, so that each steel pole has a direct cable connection with the generator. (3) The third method is now being adopted for the latest furnaces of this type. The current is conducted to the carbon electrode in a manner similar to method 2, but whereas in methods 1 and 2 the bottom steel electrodes are insulated from the furnace body, in this method the steel electrodes are electrically connected to the furnace body; also the conductors are bus bars instead of cables attached to the steel shell of the furnace. These bus bars are arranged symmetrically around the furnace. This arrangement is shown in figure 22.^a The advantages of this arrangement are: A better agitation of the bath due to the arc circling around the periphery of the carbon electrode, greater durability of roof and lining, a saving of 10 per cent of energy consumption, the use of copper bus bars instead of cables, lack of current interruptions due to rupture of the arc, and reduction of electrode consumption.

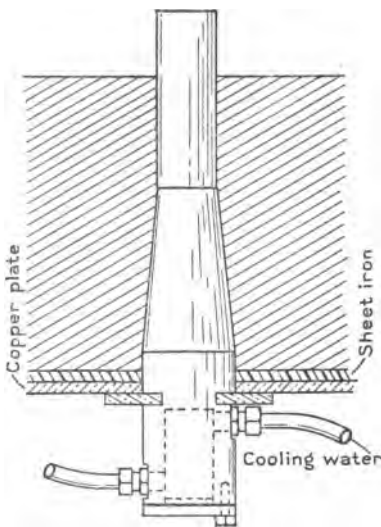


FIGURE 29.—Section of water-cooled steel electrode used in Girod furnace.

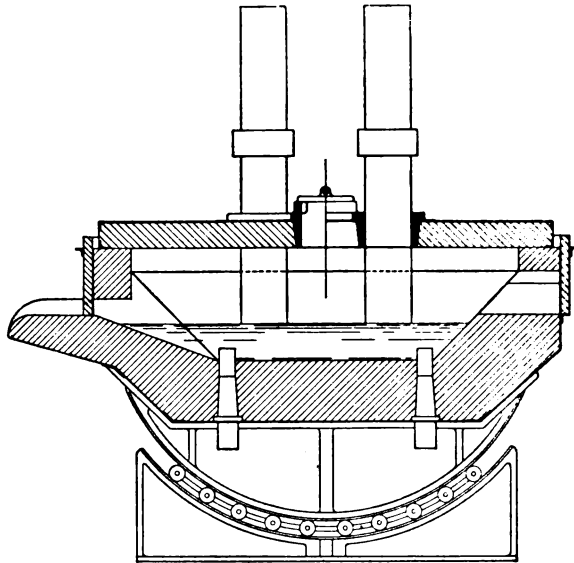
The cost of the metallic parts of a 2.5-ton Girod furnace, including electrode regulators, measuring instruments, tilting device, and conductors from the furnace to a dynamo or transformer near the furnace room, not including transforming or generating machinery and license fee, is estimated at about \$3,000. The cost of a plant consisting of one 2.5-ton furnace for regular running and one furnace for reserve, with all appliances and smelter building, but without dynamo or transformer, is estimated to be approximately \$40,000 to \$50,000. The license fee is not included in this estimate.

THREE-PHASE GIROD FURNACE.

A section of the three-phase 10 to 25 ton Girod furnace is shown in figure 30. There are four upper carbon electrodes 14 inches in

^a Mueller, A., The manufacture of steel in the Girod electric furnace: *Metall. Chem. Eng.*, vol. 9, 1911, p. 581.

diameter, which with their connections constitute the essential difference between the single-phase and the three-phase furnace. On a three-phase circuit the Girod furnace is connected by the star con-



SECTION A-A

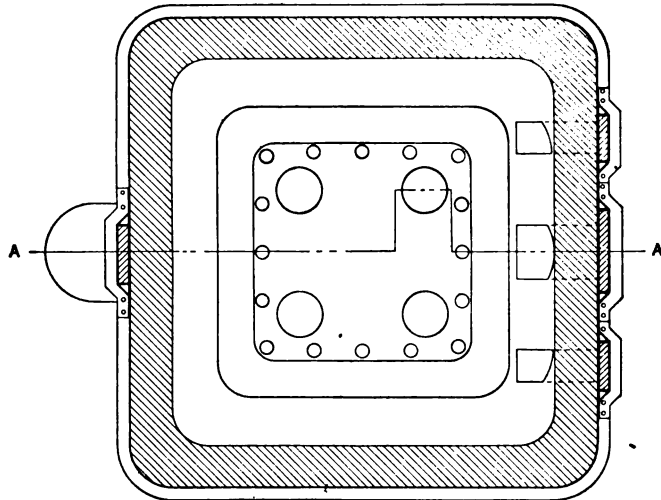


FIGURE 30.—Plan and elevation of 10 to 12.5 ton, three-phase Girod steel furnace, Ugine, France.

nection. Two of the carbon electrodes are each connected to a phase. The other two are connected in parallel with the third phase, while the hearth is connected so as to form the neutral point of the system.

There are 16 bottom electrodes of steel. The 12-ton furnace uses from 1,000 to 1,200 kw., supplied by a 25-cycle current at 70 to 75 volts.

The interior of the hearth of the 12-ton furnace is 4 feet square at the bottom, widening out to 10 feet at the top. The silica brick roof is set 3 feet 9 inches above the hearth of dolomite. The bottom is tamped in to a depth of 20 inches.

The furnace, with the exception that there are three charging doors at the rear instead of one, is similar to the single-phase 2.5-ton furnace. The four electrode holders are set two on a side, arranged as in the smaller furnaces.

The metallic parts of a 10 to 12.5 ton three-phase Girod furnace, including regulators for the electrodes, measuring instruments, tilting mechanism, and conductors from the furnace to a dynamo or transformer near the furnace room, but not transforming or generating machinery and license fee, cost about \$6,000. A plant with one 12-ton furnace in reserve, including building, but without dynamo or transformer, is estimated to cost, license fee excluded, about \$60,000 to \$100,000.

THE STASSANO FURNACE.

The Stassano* electric steel furnace differs from other electric steel furnaces in that the electrical current does not pass through the metal or slag. All heating is by radiation from three horizontal arcs. The power consumption of the Stassano furnace is somewhat higher than that of some others, so that its use is limited to furnaces having a capacity of not more than 2 tons for making high-grade small steel castings. At present there are 16 Stassano furnaces in operation in sizes up to 2 tons capacity and one is in course of construction.

The Stassano furnace (figs. 19 and 20) has a circular hearth with a cylindrical melting chamber above. In the 1-ton furnace (figs. 19 and 20) the hearth is about 3 feet in diameter, with the roof 3 feet 6 inches above it. The furnace is incased in a steel shell with one door opposite the pouring spout. There are also openings for the three electrodes which project toward the center of the melting chamber. The movement of each electrode is controlled by a hydraulic piston. The source of power is connected by a rod with the end of each electrode, each of which is connected to a phase of a three-phase system. Each electrode holder and electrode are surrounded by a water jacket. In the 2-ton furnace carbon electrodes 3 inches in diameter and 4 or 5 feet long are used. The lining of the hearth is dolomite and tar, but the roof and sides are magnesite

* Stassano, E., The application of the electric furnace to siderurgy: Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 63.

brick. The external dimensions of the 1-ton furnace are about 10 feet by 8 feet; the 2-ton furnace is 10 feet high and 10 feet in diameter.

The early type of Stassano furnace had a chamber beneath it (fig. 19) which contained a rotating mechanism for agitating the steel bath. This feature has not proved of great value, and is now abandoned in the more recent furnaces, which, like the Girod furnace, are set on rollers. Some furnaces recently erected at Newcastle on Tyne, England, by the Electroflex Steel Co. have a combination of the rotating and tilting motion.

A 1-ton Stassano furnace requires 200 kw. supplied by a 3-phase at 110 volts, having a frequency of 25 cycles. A 2-ton furnace uses about 400 kw. at 120 volts.

THE KELLER FURNACE.

The Keller^a furnace (fig. 23) is very similar to the Girod furnace, as it has a conducting hearth of iron rods embedded in a refractory material. This type of furnace was the first to be used as a super-refining agent for steel that had been made by the old-established methods. There are in Europe three Keller furnaces, the capacities of which are 1 to 8 tons.

The Keller furnace consists of a conducting hearth surrounded by a steel water jacket, with a silica brick roof. The conducting hearth consists of iron bars 1 to 1½ inches diameter, set vertically 1 inch apart in an iron plate. These bars are surrounded by a mixture of magnesite and tar rammed in while hot. The bars are good conductors of electricity when the furnace is cold, and the magnesite also becomes a conductor when hot. The small furnaces have one carbon electrode and the larger ones have four electrodes. In both types the connections are similar to those of the Girod furnace. A feature of the Keller furnace plant, devised before the day of continuous feeding of electrodes, is the revolving arms, with extra electrodes for quick charging (fig. 23). An 8-ton Keller furnace is operated with 750 kw.

THE GRÖNWALL FURNACE.

In general external appearance the Grönwall^b or Electro-Metals furnace (figs. 24 and 25) resembles the single-phase Héroult furnace. However, it uses two-phase current and has a conducting hearth. A noteworthy feature of this design is the steadiness of the load on the power line when cold scrap is worked, as the arcs are not connected in series. There are four Grönwall furnaces in operation.

^a Keller, C. A., A contribution to the study of electric furnaces as applied to the manufacture of iron and steel: Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 87.

^b Robertson, T. D., The Grönwall steel refining furnace: Metall. Chem. Eng., vol. 9, 1911, p. 573.

The current may be directly supplied as two-phase or transformed from three-phase to two-phase by the Scott connection of transformers. In the Electro-Metals furnace at Sheffield there are two 12-inch carbon electrodes projecting into the furnace through the roof, each of which is connected to a phase, the conducting hearth forming the neutral of the system. This conducting hearth consists of carbon paste rammed on the steel bottom of the furnace to a depth of 4 inches, over which is placed a mixture of dolomite and tar to a depth of 10 inches. In figures 24 and 25 the neutral point is a carbon block, but, as stated above, this has been recently changed at the Sheffield furnace. By this connection half of the power goes through one electrode and half through the other, each being independent of the other.

The furnace proper is rectangular in shape, like the Héroult furnace, the exterior being 6 feet 6 inches wide by 8 feet 10 inches long by 4 feet 11 inches high. There are two doors at the sides and a pouring spout, with the electrode holders, at the rear. The holders are of manganese steel, held tightly around the electrodes by wedges. At first, as shown in figure 25, current was led to the electrodes through these holders, but at present the electrodes are electrically connected with cables and two copper plates, $\frac{1}{4}$ inch thick and 6 inches wide, shaped so as to pass around the electrode. The furnace is tilted by hand by means of a screw device. Electrodes are adjusted by hand, although there is no technical reason why an automatic regulator should not be used. The lining of the sides is magnesite brick and the roof is silica brick. The use of water jackets around the electrodes has been discontinued. As now operated there is no water cooling of any part of the furnace. The furnace is set upon a concrete foundation, being tilted on the usual curved bars. The power required is 500 kw., the frequency of the current being 25 cycles; the voltage of the two carbon electrodes is 105, while between each electrode and the neutral point, the hearth, the voltage is about 75.

THE NATHUSIUS FURNACE.

The Nathusius^a furnace (fig. 26) is three-phase and combines heating from above by arcs with resistance heating from below in the steel bath for the purpose of decreasing local heating by the arcs. The furnace is circular in form and is incased in a steel shell. There are three water-cooled carbon electrodes which project through the roof into the furnace above the surface of the charge, and three or a multiple of three water-cooled bottom electrodes of mild steel set in the hearth. Both upper and lower electrodes are arranged in a

^a Nathusius, H., The refining of steel in the Nathusius electric furnace: Jour. Iron and Steel Inst., vol. 85, 1912, No. 1, p. 51.

triangle (p. 76). The electric connections are shown in figure 26. The electrodes are suspended by cables from overhead runways and are adjusted either automatically or by hand. On tilting, these electrodes are raised out of the furnace. The tilting mechanism is operated by an electric motor. Furnaces under 6-ton capacity are on trunnions, but with the larger sizes rollers are used. The 5-ton furnace at Friedenshütte, Germany, is normally operated with about 600 kw. Sixty-cycle current is used. The voltage between the upper electrodes is 110 volts, between the lower electrodes 10 volts, and between the upper and lower electrodes 61 volts. In addition to this 5-ton furnace the same company operates a 2 to 3 ton furnace for melting ferro-alloys.

OTHER ELECTRIC STEEL FURNACES OF THE ARC TYPE.

Several other types of arc furnaces which have no especially novel feature are in operation at the plants where they were originally designed. The Chaplet furnace, used for the manufacture of ferro-alloys, the direct production of steel from ores and scrap, is similar in principle to the Héroult furnace, but has two separate chambers with an electrode in each. Four Chaplet furnaces are in operation in France. The Anderson furnace is also similar to the Héroult furnace, but has an electromagnet beneath it for the purpose of controlling the position of the arc. Five Anderson furnaces have been built in England. The Stobie two-phase furnace is very similar to the Grönwall furnace. There has also been designed a Stobie three-phase furnace. Four Stobie furnaces are being built at Newcastle, England. One three-phase Soderburg furnace, similar to the Girod furnace, is in operation. In the design of the Harden Paragon^a furnace the bath is heated from above by arcs, and also from the sides and bottom by side plates in the lining.

INDUCTION FURNACES.

THE KJELLIN FURNACE.

The Kjellin furnace, the pioneer of induction steel-furnaces, is especially adapted to the melting of fine materials to obtain a high-grade steel. At present 9 of these furnaces are in operation, but the writer does not know of any others being erected.

The Kjellin^b furnace (fig. 21) is in reality a transformer in which the bath of molten metal forms the secondary circuit. The magnetic

^a Hårdén, J., The Paragon electric furnace and recent developments in metallurgy: *Met. and Chem. Eng.*, vol. 9, 1911, p. 595.

^b Kjellin, F. A., The Kjellin and Röschling-Rodenhauser electric furnaces: *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 173.

circuit C is built up of laminated sheet iron like the core of a transformer. The primary circuit is a coil consisting of a number of turns of insulated copper wire or tubing surrounding the magnetic circuit. The ring-shaped crucible B, made of suitable refractory materials, also surrounds the magnetic circuit, and when filled with molten metal forms the secondary circuit of the transformer. The annular crucible is supplied with covers.

If the coil be connected with the poles of an alternating current generator, the current passing through the coil excites a variable magnetic flux in the iron core, and the variation in the magnetic flux induces a current in the closed circuit formed by the molten metal in the crucible B. The ratio between the primary and secondary current is fixed by the number of turns of the primary, and the magnitude of the current in the steel is then almost the same as the primary current multiplied by the turns of the primary coil. Thus in a small furnace of this type a current of 500 volts and 280 amperes supplied to the coil induces a current of 7 volts and 24,000 amperes in the metallic bath. Before starting the furnace an iron ring must be placed in the crucible and melted down to form a bath, or the crucible must be filled with molten metal taken from another source. On continuous work it is customary to leave enough metal in the crucible to establish the bath. Kjellin furnaces have been built in sizes up to 8.5 tons capacity, requiring 750 kw.

THE RÖCHLING-RODENHAUSER FURNACE.

The Röchling-Rodenhauser furnace is an improvement of the Kjellin induction furnace designed especially for the refining of molten basic Bessemer steel and for use with currents of frequency ordinarily used in steel plants. The Kjellin furnace of 8 tons capacity required a current of not more than 5 cycles frequency or the power factor would drop below 0.6. The Röchling-Rodenhauser furnace is so designed as to have a power factor of 0.6 when using a frequency as high as 50 cycles. The furnace is the most widely used of the induction furnaces and is especially adapted to refining molten metal. Eighteen of these furnaces are already in operation and four are in course of construction.

The Röchling-Rodenhauser^a furnace (fig. 27) has a hearth of very different shape from the Kjellin furnace, as it has a distinct open hearth which no other induction furnace possesses. Both single and three-phase current furnaces are constructed, the former having two grooves in which the metal is melted and the latter three. In either

^a Rodenhauser, W., The electric furnace and electric process of steel making: Jour. Iron and Steel Inst., vol. 79, No. 1, 1909, p. 261.

furnace these grooves or heating channels, each of which corresponds to the annular hearth of the Kjellin furnace, open into a distinct open hearth, where all metallurgical operations, such as the addition of fluxes and alloys, take place. The grooves, which have a comparatively small cross section, form the secondary circuits in which the currents that heat the metal are induced.

Two side doors are provided in a single-phase furnace and three in a three-phase furnace. The furnace is set on rollers and is tilted by means of a hydraulic motor. One door has a spout for pouring. As the doors are slightly above the level of the bath, removing the slag is no more difficult than in the arc furnace, which is not the case with other induction furnaces.

In figure 27, HH represents the two legs of the transformer that are surrounded by primary coils A connected with the alternating-current circuit. Secondary currents are induced in the two closed circuits formed by the bath. These two circuits are connected by the hearth in the center so as to resemble the figure 8. The primary coils are so arranged that the induced currents have the same direction in the common part of the two circuits. The difference between this furnace and the ordinary induction furnace consists in the use of extra secondary coils BB surrounding the primary coils AA. The secondary coils are connected to metallic plates EE, covered by an electrically conducting mixture of lining material, G, which forms part of the lining of the furnace. The current from the secondaries passes through the plates, E, through the lining, G, and then through the main hearth, D.

This current used in combination with the current from the channels CC gives a better power factor than when the furnace is operated with the current from the channels only. Another advantage is that the magnetic leakage field surrounding the primary coils, which formerly had the effect of checking the primary current and decreasing the power factor, is now utilized for inducing currents in the extra secondaries.

The result is that the main hearth can be made of much larger cross section than before and that, nevertheless, even in big furnaces a good power factor can be maintained without the use of such a low frequency of the current as was necessary with the original induction furnace.

The single-phase furnace gives better satisfaction than the three-phase furnace because it is less complicated. Also, the strong circulation of the bath in the central hearth of the three-phase furnace causes great wear on the lining.

Röchling-Rodenhauser furnaces have been built in various sizes. The 8 to 10 ton single-phase furnace at Volklingen, Germany, takes 600 kw., supplied by a current of 4,000 to 5,000 volts with a fre-

quency of 25 cycles. The 2 to 3 ton 3-phase furnace at the same place takes from 200 to 250 kw., supplied by a 400-volt current with a frequency of 50 cycles. The power load of these furnaces is very steady. The 2 to 3 ton furnace recently installed at Landsdowne, Pa., requires 300 kw., furnished by a 25-cycle current at 480 volts.

OTHER INDUCTION STEEL FURNACES.

The Frick furnace is very similar to the Kjellin furnace, differing from it only in slight details of design. The primary windings are placed above and below the annular ring instead of within it. One 12-ton Frick furnace is in operation in Germany.

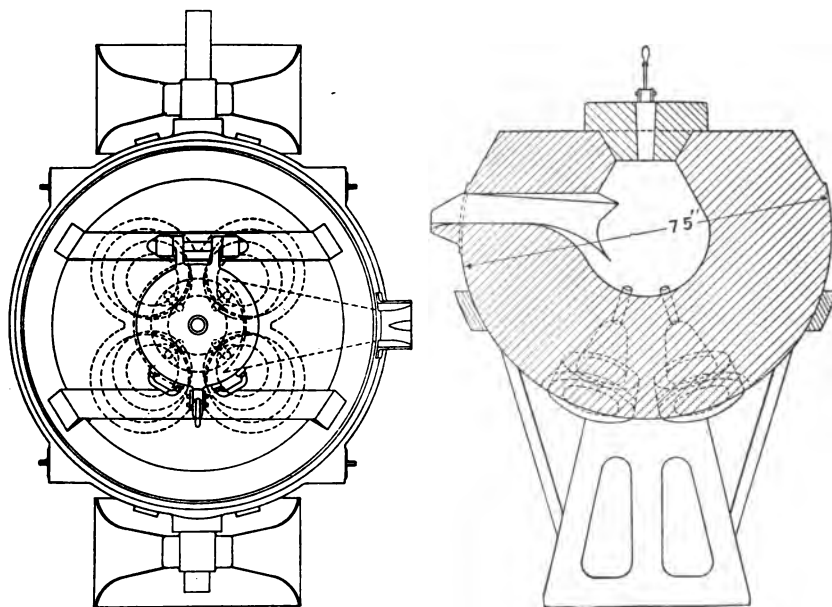


FIGURE 31.—Plan and elevation of Hering resistance furnace.

The Hiorth furnace is a further development of the Kjellin furnace, differing only in that the bath lies in two annular grooves instead of one, thus surrounding both sides of the primary coil. One tilting Hiorth furnace has been erected in Norway.

AN ELECTRICAL RESISTANCE FURNACE ADAPTABLE TO STEEL MANUFACTURE.

A resistance furnace based upon the "pinch effect" has been designed and operated on an experimental scale by Hering.^a Although this furnace (fig. 31) is not in commercial operation as

^a Hering, C., A new type of electric furnace: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 255.

yet for the production of steel, several are in the course of construction for this purpose, and small ones have been shown in operation. The following is a brief description of the principle on which this resistance furnace operates.

The "pinch phenomenon" is the local contraction of cross section of a liquid resistor through which electric current is passing and in which heat is being generated. In open channels this contraction or pinching often results in complete rupture, thereby limiting the temperature. This contraction is caused by an electromagnetic force that acts from the circumference to the center of the conductor and in a direction perpendicular to the axis.

If a conductor consists of a column of liquid metal in a vertical hole in a nonconducting material closed at the bottom by the electrode and opening at the top into the bath of metal in the hearth, then this force acts horizontally, and perpendicularly to the axis along the whole length, which results in an axial force that causes the liquid metal to flow out of the center of the open end. In the Hering furnace such a column is made the resistor in which the desired heat is generated by passing the current through the column by means of an electrode at the bottom. Two such resistors are placed in the bottom of a furnace of any desired shape for single-phase current and three for three-phase current. As this "pinch effect" can not now rupture the circuit, it expels the heated liquid rapidly from the holes and forces it against the blanket of slag, thereby continually renewing the surface exposed to the slag action, while the cooler liquid in the bottom of the hearth is sucked down into the hole near the circumference to be in turn heated and immediately expelled. The ejecting force increases as the square of the current and diminishes with an increase in the cross section of the conductor.

There is an active and systematic circulation of the liquid bath, and the temperature is very evenly distributed throughout the bath. As far as is known now, there is no temperature limit except that which causes failure of the refractory lining. The electrodes may be made of metal, and no adjustment of them is necessary; they are not consumed. The hottest liquid is in the center of the resistors. The rapid flow is not along the walls of the hole but in the center. The power factor can be made very high, and ordinary frequencies may be used.

The furnace can be operated with direct or alternating current of one, two, or three phase. The transformers in the latter case are attached directly to the bottom of the furnace, as the amperage is very high.

ELECTRIC STEEL MANUFACTURING PRACTICE.

There are two courses of procedure for steel manufacture in which the electric furnace has been used: Cold scrap iron and steel of either inferior or high-grade quality is melted and refined in an electric furnace with the production of steel of the highest grade equal to the best crucible steel; and molten steel, the product of either the acid or basic converters, or of the acid or basic open-hearth furnaces, is superrefined, or made into alloy steel, in an electric furnace.

MANUFACTURE OF STEEL FROM SCRAP IRON AND STEEL IN THE ELECTRIC FURNACE.

SOCIÉTÉ ÉLECTRO-MÉTALLURGIQUE FRANÇAISE, LA PRAZ, SAVOIE, FRANCE.

The plant of the Société Électro-Métallurgique Française, at La Praz, Savoy, France, has passed through many phases of the use of the electric furnace in metallurgical operations. The first product produced here by Héroult was aluminum, which was followed by calcium carbide, ferro-alloys, electrodes, and steel. To-day the plant uses 7,500 kilowatts for the production of aluminum, steel, and electrodes.

DESCRIPTION OF PLANT.

The principal products of the 2.5 to 3 ton Héroult furnace (fig. 18) in operation at La Praz are high carbon and alloy tool steels. The furnace is the original Héroult furnace examined by the Canadian commission. It differs from the furnace at Braintree in that the lining is thinner, giving a more shallow hearth. The hearth is ground dolomite and tar, the sides are magnesite brick, and the roof is silica brick. The two carbon electrodes are 14 inches square. The holders are of somewhat more simple design than those in the Braintree furnace (p. 94). They consist of two clamps of steel held together by a pin, being tightened by a wedge. Current is brought in by cables attached to pins in the ends of the electrodes and to the electrode holders. Electrodes are not threaded for continuous feeding. The holders and ports in the roof are water cooled. The electrodes may be regulated either by hand or automatically. Power is supplied by a single-phase alternator directly connected to the furnace about 30 feet away. The power required is 300 kw., and is supplied by a 33-cycle single-phase current, at 100 to 110 volts, giving 50 to 55 volts on each electrode.

PRACTICE AT PLANT.

The furnaces are charged with low-carbon steel or wrought-iron turnings and a part of the first refining flux. As the melting proceeds, all of the first flux, consisting of iron ore and lime, is added to remove the phosphorus. When this is accomplished the slag is

completely removed from the furnace and a flux of lime added to remove the sulphur. The slag is completely deoxidized for the removal of sulphur by the addition of coke dust. The metal is finally carburized to the desired point by addition of carburite, and then poured into ingots.

The results of operations covering one week are given below. The number of heats is probably somewhat greater than the usual number, as at the time of the writer's visit the average was three in 24 hours. A noteworthy reduction of power consumption has been made over that of 1904, as the figure of December 24, 1911, was 528 kilowatt-hours per ton as compared to 840 kilowatt-hours per ton, of 1 per cent carbon steel, refined with two slags, for 1904, a reduction of 312 kilowatt-hours, or 37.1 per cent decrease. The electrode consumption of 18 kg. (39.6 pounds) in 1904 would probably approximately hold for to-day, as the methods have not been changed materially in that respect. The roof had stood 107 heats up to the week mentioned.

Results of operation of furnace for week ended Dec. 24, 1911.

Hours worked.....	126
Number of heats.....	26
Average hours per heat, including all repairs and charging.....	4 hours, 51 minutes.
Shortest heat.....	3 hours, 10 minutes.
Total weight teemed.....	66 tons.
Lowest power consumption, per ton, at furnace.....	459.2 kilowatt-hours.
Average power for week at furnace, per ton.....	528 kilowatt-hours.
Scrap, per cent.....	3
Clear ingots, per cent.....	93
Loss by oxidation, per cent.....	4

FINAL FORM OF PRODUCT.

The ingots cast at the furnace are reheated and reduced in rolls and hammered to 1 inch to $\frac{1}{2}$ inch rods of circular or square cross section. The product is sold in this form. Samples of steels of the following composition were taken by the writer:

Tool steels produced in Héroult furnace, La Praz, France, 1912.

	No. of sample.			
	1	2	3	4
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Carbon.....	0.904	1.034	1.196	0.990
Silicon.....	.323	.201	.170	.700
Manganese.....	.236	.273	.210	.211
Sulphur.....	.008	.008	.009	.024
Phosphorus.....	.004	.006	.012	.007
Tungsten.....				19.407
Chromium.....				6.717

PLANT OF MESSRS. VICKERS, LTD., SHEFFIELD, ENGLAND.

A 2.5 to 3 ton single-phase Héroult furnace, similar to the furnace at Braintree, is being used in the manufacture of steel of crucible grade from cold scrap iron and steel at the River Don plant of Messrs. Vickers, Ltd., Sheffield, England. A new 8-ton three-phase Héroult furnace is being constructed, which will be directly connected to the main power circuit of the works and will take about 1,200 kw. The lining of rammed dolomite and tar, both on sides and bottom, will be thinner than usual, so that the nominal capacity will be 10 instead of 8 tons. The electrodes are to be supported by holders projecting over the roof from the sides. Charging doors will be placed in the rear instead of at the side, with the pouring spout at the front. The furnace will be set close to the edge of the working floor instead of 10 feet back, as is the case with the present 2.5-ton furnace. The erection of an electrode plant is also being considered.

DESCRIPTION OF PLANT.

The electric furnace, generator, and laboratory are placed in a steel frame building about 60 feet square, covered with corrugated iron. In one corner, at the rear of the furnace on the ground level, is the generator room. Above, on the furnace level, are the office and laboratory. The furnace is set on a concrete foundation extending about 6 feet above the ground with a steel platform about 20 feet square around it. The furnace is at the rear of this platform, leaving an open space of about 10 feet in front, in which there is a door that can be lifted for setting the ladle beneath the spout in pouring. The instrument board and Thury regulators are back of the furnace. In front of the platform is a pit 5 feet long, 20 feet wide, and 4 feet deep, where ingot molds are set for teeming. The remaining floor space is devoted to repairing and heating ladles, storing ingots, and piling scrap. There is an electric crane in front of the furnace platform.

The furnace differs in some details from the Braintree furnace. The bottom consists of a $4\frac{1}{2}$ -inch layer of magnesite brick laid on edge over the steel shell, above which tar and dolomite are rammed in to a depth of 7 inches, making a hearth 11.5 inches thick. This slopes up to a line on the sides above the slag line. When the furnace was first operated the walls were built of magnesite bricks laid to give a thickness of 9 inches. Because of the bricks spalling they have now been laid so as to give a wall $4\frac{1}{2}$ inches thick. This, incidentally, has increased the capacity, so that the average charge is now 3 tons instead of 2.5 tons. The roof is silica brick. Carbon electrodes 16 inches in diameter and 4 feet long, and threaded for continuous feeding, are used. The electrodes are supported by a more simple holder than that of the Braintree furnace, consisting of bronze clamps surround-

ing the electrode, tightened in front by a horizontal bolt. The holder is similar to the older type, except for this feature, and that it is made entirely of manganese steel instead of copper and is not used for conducting current. Current is conducted by means of copper bus bars 4 inches wide and $\frac{7}{8}$ inch thick from cables at the rear of and beneath the furnace to the bronze holders around the electrodes. The furnace is tilted by means of a screw device that is operated by an electric motor. The electrode holders are water cooled, and there are copper water jackets around the electrode openings in the roof.

Power is delivered to the furnace about 20 feet from the generator by a single-phase alternating-current generator of 600 kw. capacity, operating at 100 to 110 volts and a frequency of 25 cycles. The alternator is directly connected to a 3-phase motor. The power is received from the main power plant of the works at 2,200 volts. The current consumption charged to the furnace is that shown on the primary side of the main 3-phase line, so that it includes all transforming and other losses. In the generator room there are also a power-factor meter and other meters on the secondary side. Current is conducted to the furnace bus bars by insulated copper cables. At the rear of the furnace are the Thury regulators, a wattmeter, a voltmeter on both electrodes and on each single electrode.

PRACTICE AT PLANT.

The electric-furnace operation consists in making high-grade steel for various purposes, such as tools and armament, out of scrap iron and steel turnings, some of which are impure ordinary carbon steel, while others contain valuable proportions of tungsten, chromium, and other expensive alloys. The furnace is operated at 100 volts, 50 volts per electrode, with a power consumption of 400 to 550 kw. For 15 to 45 minutes after charging, the electrodes are regulated by hand, but as soon as a pool of molten metal has formed in the bottom the Thury regulators are thrown in. The constant breaking of the circuit is very noticeable before the charge is completely melted. Sometimes one and sometimes two refining fluxes are used, depending upon the impurities present. The first flux is generally 40 kg. (88 pounds) of lime, 30 kg. (66 pounds) of hematite, and 20 kg. (44 pounds) of fluorspar. The second flux consists of 40 kg. (88 pounds) of lime and 20 kg. (44 pounds) of fluorspar, and is added only after complete dephosphorization by the first slag. To deoxidize the second slag for removing sulphur, coke dust is added. Sometimes pig iron, ferrosilicon, ferromanganese, and sand are used at the end of a run. Aluminum is thrown in the ladle bottom before pouring. A 3-ton ladle is used for teeming and is preheated before using. Part of the metal from one heat is cast by bottom casting

into five 500-kilogram (1,100-pound) ingots, the balance of the charge being cast by top casting into 100-kilogram (220-pound) ingots. Of the metal charged, about 85 per cent is recovered in good ingots, 5 per cent in poor ingots, 3 per cent in scrap, and 7 per cent is oxidized.

Working with high-speed tool-steel scrap, the method of running a charge was as follows:

Results from a single charge.

Charge, steel scrap, tool-steel scrap, 2,800 kg. (6,160 pounds).

Slag added, 40 kg. (88 pounds) lime and 20 kg. (44 pounds) fluorspar.

No slag skimmed.

Additions at end, 6 kg. (13.2 pounds) of a deoxidizing mixture, 2 kg. (4.4 pounds) ferrosilicon, 0.5 kg. (1.1 pounds) aluminum. No coke dust added.

A gray slag.

Time, hours	3½
Total power on primary meter, kw.....	1,580
Kilowatt-hours per metric ton of sound ingots.....	542
Average power, kw.....	513
Composition of product:	

	Per cent.
Carbon	0.60
Tungsten	4.57
Chromium	5.92
Manganese	.20
Silicon	.07
Phosphorus	.012
Sulphur	.052

It will be noted that the power consumption is higher in this case for an operation with one slag than at La Praz with two slags, but this may be explained by the fact that power charged to the furnace at Sheffield is read on the primary circuit and includes all losses; also results are figured to tons of sound ingots. The power factor averages about 0.85, but varied over a period of a year from 0.79 to 0.95.

The average length of a heat is 4 hours, but at times as many as 6 heats per 24 hours have been made, including charging and pouring. The electrode cost is about 75 cents per metric ton which, with electrodes at 4 cents a pound, gives a consumption of 18.7 pounds per ton. It takes about 20 minutes to add a new 4-foot section to an electrode already in the furnace. The electrodes wear uniformly around the holder and at joints, but if a joint is not properly made the electrode tends to wear to a point, and then must be broken off at the end. If a piece falls into the bath it is necessary to shut off the power while the piece is being removed. Between heats the power is off for 10 to 15 minutes. The furnace lining is then fettled with dolomite. For a period of six months' continuous operation the roofs lasted on the average 78 heats, but some have lasted as long as 120 heats.

Advocates of the conducting-hearth type of furnace have stated that it would be very difficult to melt out the charge of a Héroult furnace if frozen, but this does not appear to be the case. At this plant a 2.5-ton charge was frozen, owing to a breakdown of the generator, and stood cold for four days. The charge was melted in 12 hours with a power expenditure of 1,950 kilowatt-hours per ton. During the melting 0.2 ton more of metal was added, making 2.7 tons in the furnace. Of this 2.525 tons were tapped. For about $1\frac{1}{2}$ hours the power on the furnace was only 250 kw. in order to heat the roof gradually.

For the complete operation of this plant seven men and two boys are necessary, as follows: A superintendent, assistant superintendent, melter, assistant melter, ladle man, floor man, one general laborer, and two boys in the laboratory.

LAKES & ELLIOT FOUNDRY, BRAINTREE, ENGLAND.

At the foundry of Lakes & Elliot, Braintree, England, a 2.5-ton, single-phase Héroult furnace is used in making steel of low carbon content for small castings for the automobile trade.

DESCRIPTION OF PLANT.

The furnace, which has already been described (p. 74), is placed in the main casting building of the foundry, close to one wall. At the rear in an adjoining building is a separate power plant for the furnace, consisting of a gas producer, a Westinghouse gas engine, and a single-phase 300-kilowatt generator which delivers a 25-cycle current to the furnace about 20 feet away at 110 volts. A 5-ton overhead electric crane is used for pouring and other purposes.

PRACTICE AT PLANT.

The scrap iron and steel used consists of a mixture of horseshoes, castings, foundry scrap, boiler punchings, and miscellaneous heavy steel scrap of small dimensions. Some pig iron is also used, which is placed between the electrodes and is claimed to assist in the steady operation of the furnace. The steel charged contains from 0.05 to 0.14 per cent carbon. A part of the iron ore and lime that form the dephosphorizing slag is laid upon the floor of the furnace before charging the scrap. This seems to protect the hearth from corrosion. The remainder of this flux is added during the melting period. After skimming the slag a desulphurizing flux of lime and fluorspar is used. The carbon is reduced to 0.06 per cent and recarburization is accomplished by means of powdered carbon. Some ferromanganese, ferrosilicon, ferrotitanium, and aluminum are added at the end of the run or upon pouring. The charge is poured into a 2.5-ton ladle,

from which it is teemed into small shanks for casting into the molds, thus eliminating slag. The electrodes are regulated by hand for 1½ hours after starting the furnace until the bath is partly melted.

Owing to the thick lining used, charges of not over 2 tons can be made, and the average melt is not over 1.5 tons. The melt takes 4.5 to 11 hours, with an average of about 6 hours, depending upon the nature of the charge. Definite figures as to power consumption could not be obtained, but with 300 kw. on the furnace, it is probably from 800 to 1,000 kw. per ton. One charge is run on the day shift and one on the night shift, the furnace standing idle in the meantime. By this arrangement the furnace is empty 2 to 4 hours between melts.

The electrode consumption could not be obtained definitely, but two 5.5-foot sections of 14-inch electrodes will last for 14 melts. These sections weigh about 600 pounds, which with an average charge of 1.5 tons would give a consumption of 25.5 kg. (56 pounds) per ton. The cost of electrodes is about 3.95 cents per kg. (1.8 cents per pound) f. o. b. plant, or 4.95 cents per kg. (2.25 cents per pound) laid down at Braintree.

Considerable difficulty has been experienced at Braintree in getting a durable roof. Several kinds of brick, such as silica, bauxite, magnesite, and a mixture of magnesite and bauxite, have been tried. At present bauxite brick is being used. The roof lasts 14 to 28 heats, depending upon the nature of the charge. The walls are also attacked by the arc breaking against the sides after the charge has been melted. This, however, does not cause so much trouble when a slag is on the bath. Whenever the roof gives out the furnace is entirely relined.

In addition to these difficulties, the Thury regulators did not appear to be giving satisfaction, because they were continually getting out of adjustment. This was the only plant visited where this trouble was mentioned. It was stated that in building a new furnace automatic regulators would not be used, as there was no saving in labor and an increase in first cost by their use.

The effect of producing low-carbon steel upon the working of the furnace was plainly visible at this plant. At the Sheffield plant, which was operating almost entirely on medium or high carbon steel, the power consumption, electrode consumption, and wear on the lining were much lower than at this plant. This was due chiefly to the low-carbon product requiring a longer period in the furnace, and also because in making steel castings the temperature of the metal must be higher than in casting ingots. The power consumption is increased by the cooling of the furnace between melts. The electrode consumption was also probably increased because the holder clamps wore into the electrode, and the worn part spalled and burned to a point when it got down into the furnace. It will be recalled that this

holder is of different type than that used on the older Héroult furnaces. The difficulty with the lining and roof was increased by the necessarily high temperature of the slag and metal. Also the writer believes that less difficulty in this respect would be experienced if the hearth was made more shallow and the lining thinner. This would prevent the "flaming arc," as the electrodes would be farther from the sides. The arc would be more concentrated beneath the electrode and less heat would be directly radiated to the roof.

PRODUCTS.

As previously stated, the product of this furnace is a soft steel containing on an average 0.14 per cent of carbon, the content varying between 0.10 per cent and 0.20 per cent. The manganese and silicon are varied according to the use to which the steel is to be put. Sulphur and phosphorus are kept below 0.03 per cent. The steel produced is equal in all respects to crucible steel manufactured at the same plant.

GIROD STEEL PLANT, UGINE, FRANCE.

The Girod steel plant at UGINE, France, is the first and largest complete steel plant built that uses electric power entirely for furnace purposes. In 1898 Girod began experimental work on the manufacture of ferro-alloys with a small 20-kilowatt furnace. To-day this manufacture has developed into an industry with an annual production valued at about \$3,000,000 and using about 22,000 kw. for electric-furnace purposes during periods of high water in the streams. As previously stated, from the electric ferro-alloy furnace was developed the Girod electric furnace. The steel works were erected in 1909, and because of their uniqueness will be described in detail.

LOCATION.

The plant is situated in a place so remote from supplies of coal and raw material that only a product in which one of the chief requisites to commercial success was cheap power could be manufactured profitably there. UGINE is on the line of the Paris, Lyon & Marseilles Railroad between Annecy and Albertville, about 25 miles from Champéry. The plant is about half a mile from the main line, with which it is connected by an electric tramway for freight haulage, which is operated by the Girod company. The buildings are on the north bank of the River Arly.

POWER SUPPLY.

The total capacity of all the Girod power plants is about 28,000 kw., of which about 6,000 kw. is used at the steel plant. There is a possible development of 30,000 kw. more, a total of 58,000 kw. Not all

of this is primary power, however; that is, power 24 hours a day, 365 days a year. The average cost of power from all sources is about \$18.66 per kilowatt-year, or 0.2 cent per kilowatt-hour, based on a flat rate. There are three separate power plants, two on the Arly River and one on the Bonnant River. Some power is leased at times of low water. The power is delivered at the steel works at 45,000 volts and is stepped down to the desired voltage.

DESCRIPTION OF PLANT.

All buildings are of stone and cement construction. The stock house at the rear of the furnace house is 58 by 460 feet. It is used for the receipt and storage of steel scrap and furnace materials of all kinds. There are overhead traveling cranes and lifting magnetic cranes, used to unload carloads of scrap iron and steel.

The furnace and casting house is 70 by 550 feet. In it there are two 10 to 12.5 ton 3-phase Girod furnaces (fig. 30) and three 2.5 to 3 ton single-phase furnaces (fig. 28), which have already been described (pp. 77-78). The three small furnaces are operated with 300 kw. supplied by a 25-cycle current at 60 to 65 volts. The larger ones take 1,000 to 1,200 kw., using a 70 to 75 volt, 25-cycle current. The furnaces are placed on a concrete foundation about 6 feet above the ground floor. The working floor of the furnace is concrete, and the furnaces are set at its edge as in an open-hearth plant. The plant is arranged for a production up to 200 tons a day. One of the smaller furnaces is used entirely for small casting work. The other two produce high-priced alloy steels. The two 10 to 12.5 ton furnaces are used for making carbon steels and the more common alloy steels. In front of the furnace platform are casting pits, one end being used entirely for foundry work. The furnace house is provided with two 2-ton traveling electric cranes and two 12-ton traveling electric cranes.

The rolling-mill building is 90 by 250 feet and contains 1 train of three-high rolls capable of rolling ingots 20 inches in diameter and 400 kg. (880 pounds) in weight to rods 5 inches in diameter. Another three-high mill rolls rods 13 inches in diameter to smaller rods of round, square, and other shapes. The trains are driven by a 600-kw. 3-phase motor. This building also contains 2 producer-gas fired furnaces for preheating ingots.

A large forging shop 70 by 250 feet contains 9 hammers operated by compressed air. The largest hammer weighs 5,000 kg. (11,000 pounds) and the others weigh from 1,000 kg. (2,200 pounds) to 100 kg. (220 pounds). A second forging shop contains one 1,000-ton forging press, 1 forging hammer weighing 10 tons, and 3 stamps with falling hammers, weighing, respectively, 3 tons, 2 tons, and

1 ton. In the first shop rough forgings are made, primarily for automobile machinery, shaftings, gears, tool-steel rods and projectiles.

A tempering shop 50 by 200 feet contains furnaces for tempering, annealing, and hardening large forged or cast-steel pieces. An annealing shop 33 by 67 feet contains several furnaces.

The steel foundry contains a carpenter shop for making patterns, a sand-separating plant, molding machines, molding frames, drying and heating ovens, and a sand-blast jet for cleaning finished castings. The foundry is able to produce 10 tons of steel castings daily, but castings weighing 20 tons have been cast.

The power house at the plant, which is 50 by 115 feet, is equipped with 4 electrically driven compressors and 2 rotary converters.

The warehouse for finished products is 50 by 325 feet, and is made especially large because of the variety of products, and also because it is necessary to carry common shapes over in stock during low-water periods.

In addition to the buildings mentioned there are a furnace-transformer house, a sand-storage house, a drying-oven house, an assembly shop, a machine shop, a storage house for the rolling mills, a pattern warehouse, general stores, a well-equipped physical and chemical laboratory, and an office building.

REFINING PRACTICE.

The refining of cold scrap at the Ugine plant^a is typical of electric-furnace practice in work of this nature, and is here described in detail. Almost any grade of scrap steel or scrap wrought iron is charged into the Ugine furnaces, the mean proportions of carbon, silicon, manganese, sulphur, and phosphorus in the charge being as follows:

Average proportions of carbon, silicon, manganese, sulphur, and phosphorus in charge.

	Per cent.
Carbon	0.3 to 0.4
Silicon	.1 to .3
Manganese	.6 to .8
Sulphur	.07 to .120
Phosphorus	.07 to .12

The refining operation can be divided into two periods, the oxidation period, in which phosphorus is removed from the metal, and the deoxidation period, with the elimination of sulphur.

After the furnace has been charged with cold scrap, an oxidizing flux of lime and iron ore is added. The proportions vary with the charge and product desired, but a mixture commonly used is 80 kg.

^a Girod, P., Studies in the electrometallurgy of ferro-alloys and steels: Trans. Faraday Soc., vol. 6, 1911, p. 172; The electric steel furnace in foundry practice: Metall. Chem. Eng., vol. 10, 1912, p. 663; Borchers, W., Electric smelting with the Girod furnace: Trans. Am. Inst. Min. Eng., vol. 41, 1910, p. 120.

(176 pounds) of lime and from 220 to 250 kg. (485 to 550 pounds) of iron ore. Oxidation of the impurities in the metal begins as soon as the scrap becomes pasty, so that by the time the charge is completely melted the carbon, silicon, manganese, and phosphorus contents of the metal are usually reduced to less than 0.1 per cent each. If this is not the case, the slag is withdrawn by tilting the furnace backwards and another flux of the same composition is added. To cleanse the bath of the last traces of the phosphorous-bearing slag and any phosphorus in the metal, lime is added and the resulting slag removed. This is repeated as often as may be necessary. In this first period the temperature is gradually increased, but the oxidation takes place largely at a low temperature.

When the oxidizing and cleansing slag has been removed, the first deoxidation of the bath is effected by adding reducing agents such as ferrosilicon or ferromanganese, but these alloys are added in such a proportion as not to remain in the bath, serving merely as deoxidizing agents. If a high-carbon steel is to be made, recarburization takes place at this point.

The bath is then covered with a flux consisting of about 5 parts lime, 1 part silica sand, 1 part fluorspar, and a little petroleum coke. In this period the furnace must be tightly closed. For proper deoxidation and desulphurization of the bath all iron oxide in the slag must be reduced. This is done with petroleum coke and deoxidizing alloys such as ferrosilicon, silicomanganese, ferrosilico-manganese-aluminum, or even silicon-aluminum. These alloys act energetically and form fluid slags which rise to the surface.

When the slag is completely deoxidized, which is shown by its being white and disintegrating to a powder in air, any desulphurization not completed in the first period is finished by the passage of the sulphur to the slag as calcium sulphide, the chemistry of the process will be described later (p. 124). Carburizing materials are added for finishing the metal, as well as other alloys to bring the steel to the desired composition. If any of the slag from the first period has been left in the furnace, the phosphorus in it will be reduced in the second period, passing into the steel. The average analysis of the final white slag is as follows:

Average composition of final slag.

	Per cent.
CaO.....	74.85
SiO ₂	13.20
FeO.....	.13
MnO.....	Traces.
Fe ₂ O ₃	None.
Al ₂ O ₃	1.75
MgO.....	4.22
P ₂ O ₅	.09
S.....	1.20

The power consumption recorded at the terminals of the furnace, including melting, refining, and finishing a charge of cold scrap is estimated by Girod at about 850 kilowatt-hours per ton of metal tapped for a 2.5 to 3 ton furnace; at 750 kilowatt-hours per ton of metal tapped for a 10 to 12 ton furnace. The power consumption varies with the charge and product required. A power factor of 0.80 to 0.88 is maintained.

The electrode consumption is stated to be 8 to 9 kg. (17.6 to 19.8 pounds) for a 2.5 to 3 ton furnace, and 8 to 10 kg. (17.6 to 22 pounds) for a 10 to 12 ton furnace. These figures are based on the use of electrodes made at the Girod plant, not designed for continuous feeding, so that this item might be considerably reduced.

Dolomite-and-tar hearths are used. The life of a lining is 90 to 100 heats for a furnace of 10 to 12 tons and about 120 heats for the 2.5 to 3 ton furnace. At the end of that time the side walls are repaired. All burned or oxidized parts of the walls are scraped. About 4 to 6 inches of the bottom is taken up and retamped with a new layer of dolomite. Care is taken to leave the bottom electrodes clear. The roof of the furnace is made of silica brick and stands on an average 50 heats for the 10 to 12 ton furnace and 70 heats for the 2.5 to 3 ton furnace.

The output of metal is 90 to 96 per cent of the charge, depending upon the nature of the metal charged.

A 2.5 to 3 ton furnace requires for operation 1 melter, 1 assistant, and 1 boy; the 10 to 12 ton furnace 1 melter, 2 assistants, and 1 boy. This does not include foremen, cranemen, ladle men, and other laborers.

PRODUCTS OF THE FURNACE.

An idea of the charges, power consumption, and wide variety of products made at Ugine may be obtained from the following record * of the operation of this furnace:

* Girod, P., op. cit., and Borchers, W., op. cit.

Operation and products of 2.5 to 3 ton Givod furnace.

Quality of steel manufactured.	Desired proportions of—					Time of starting.	Charge.				Time of pouring.	Production in Inputs of 350 kg.	Total product.	Incomplete analysis of metal obtained.							Electrical energy consumed per ton.
	C.	Si.	Mn.	Ni.	Cr.	Va.	Scrap.	Turnings.	Ore.	Lime.				C.	Si.	Mn.	P.	Ni.	Cr.	Va.	
Tool steel.....	0.90	0.30	0.30				1,800	200	100	60	11.10 a. m.	5½	1,725	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	900
Nickel-chrome steel.	.30	.20	.35	2.8	0.6		2,000	200	100	60	6.00 p. m.	5½	2,100	0.845	0.27	0.31	0.019	2.86	0.51		850
Nickel steel.....	.15	.20	.35	5.0			2,300	200	120	70	7.00 a. m.	6½	2,275	.156	.194	.380	.012	5.18			950
Nickel-chrome-vanadium steel.	.30	.15	.45	3.0	1.0	0.30	2,500	200	120	70	2.15 p. m.	7	2,450	.297	.120	.473	.010	3.22	1.06	0.23	900
Steel for shell plugs	.30	.20	.40				2,500	200	120	70	10.40 p. m.	6½	2,370	.312	.176	.420	.006	.013			900
Extra soft steel....	.10	.20	.30				2,800	300	200	70	7.00 a. m.	(a)	2,775	.097	.190	.317	.014	.012			925

a 7 castings.

FOUNDRY OF MÖNKEMÖLLER & CO., BONN, GERMANY.

At the foundry of Mönkemöller & Co., Bonn, Germany, three Stassano furnaces (figs. 19, 20) are producing low-carbon steel for high-grade steel castings and one additional Stassano furnace of 2-ton capacity is being built. The Stassano furnace in small units is better adapted to this use than to any other.

DESCRIPTION OF PLANT.

There are two 1-ton furnaces, each requiring 200 kw., supplied by a current at 110 volts with a frequency of 25 cycles, and one 2-ton furnace taking 400 kw. at 120 volts; the frequency is 25 cycles. The two small furnaces are of essentially the same construction as shown in figures 19 and 20, having the rotating mechanism. The larger one differs only in that it is set on rollers for tilting. The details of construction are as given in the description of the Stassano furnace.

FOUNDRY PRACTICE.

Scrap steel of various sizes and grades is used as a raw material. This is charged to a point directly beneath the three arcs. The desulphurizing and dephosphorizing operations are in general about the same as at other plants.

It takes about three to four hours to melt the charge and from four to five hours for complete melting and refining. The loss of heat in the water cooling of the electrodes is about 14 per cent. The electrodes consumption is very high, owing to failure to use up stumps. The repair cost is also high. The magnesite brick roof lasts from 80 to 100 heats; the whole furnace must then be relined. The power consumption is about 800 to 1,000 kilowatt-hours per ton. A power factor of from 0.85 to 0.90 is maintained. Power is supplied by a public-service corporation at a cost of about 1 cent per kilowatt-hour. The electrodes are regulated hydraulically, but not automatically, as a man watches the meter for changes all the time.

The metal is poured from the furnace into a 1 or 2 ton ladle, from which it is poured into small shanks for casting. All three furnaces are on the main casting floor of the foundry. Most of the castings made are of small size. The steel has an average content of 0.8 to 0.18 per cent carbon, 0.03 per cent sulphur, and 0.06 per cent phosphorus. Occasionally tool steels, with or without the addition of alloys, are made. The process is said to be much cheaper than the crucible method and gives a better product.

SHEFFIELD ANNEALING WORKS, SHEFFIELD, ENGLAND.

At the Sheffield Annealing Works, Sheffield, England, a 2 to 2½ ton Grönwall furnace is in use for the production of steel from scrap.

The chief product is alloy steel, which is cast into ingots and further treated at another works. The scrap steel used consists for the greater part of small pieces of tool scrap and other alloy scrap.

DESCRIPTION OF PLANT.

The furnace (figs. 24, 25) has been described. At some future time the cables to the electrodes are to be replaced by bus bars. The furnace foundation is set level with the ground, with a pit in front for teeming. The building is not well arranged for electric furnace work; the ventilation is very poor. Two annealing furnaces are adjacent to the casting pit.

The two-phase current is received from the main line of the municipal power system at 2,000 volts, with a frequency of 25 cycles, the cost being about 1 cent per kilowatt-hour. An integrating wattmeter, from which the power is charged to the furnace and from which power figures are taken, is placed on the primary side of the circuit where the line enters the building. The meter is followed in the circuit by choke coils to the two transformers. These transformers can be adjusted to give 50, 60, 70, or 80 volts on the secondary circuit. The usual voltage is 75 volts. The neutral point is connected to the bottom of the furnace and a phase is connected to each electrode. The secondary switchboard has a power-factor meter, which can be thrown into any leg, as well as a voltmeter, with the same adjustment. On the switchboard there is also one ammeter set in the neutral. On the back of the furnace there are two ammeters, one to each phase, by means of which two men regulate the electrodes.

PLANT PRACTICE.

The furnace is operated only 12 hours a day, so that more than two charges are never run in one day; also, in the wintertime, it is necessary not to draw too heavily on the power system late in the afternoon when the load on the city lines begins to get heavy. Such operation, of course, is not economical with respect to power consumption, as the furnace cools overnight. Constant change of temperature in the furnace also increases the wear on the lining and the electrodes.

At the time the furnace was inspected it was charged with steel scrap containing 0.40 per cent carbon, 1 to 2 per cent tungsten, and about 1 to 2 per cent chromium. The object of the run was melting with some refining and the casting of ingots. The charge melted in 2 hours and 15 minutes, during which period 560 kilowatt-hours per ton were expended, and was poured in 4 hours and 5 minutes, with a total expenditure of energy of 902 kilowatt-hours per ton. Thus 62 per cent of the energy consumed was used for melting. During the melting stage the amperage was held at 4,000 amperes, or about

500 kw. was on the furnace, while during the refining period this was reduced to 2,500 to 3,000 amperes and about 200 to 300 kw. Often only about 1,500 amperes are on toward the end, but at the very end the temperature is raised to keep the slag fluid. This is said to protect the roof, which seems to be more attacked during the last half hour than any other time. During the melting period a mixture composed of 100 pounds of lime, 30 pounds fluorspar, and 20 pounds of silica was added in two parts. None was added after the melting period, but as soon as the alloy was hot anthracite coal dust was used for deoxidation. In $1\frac{1}{2}$ hours the slag became white. At the end of 3 hours and 50 minutes samples of metal and slag were taken. The slag was white. Analysis of the steel showed a carbon content of 0.39 per cent. While the analysis was being made more coal was shoveled in to prevent reoxidation of the slag. Another sample of slag was taken and the charge poured into a 2.5-ton ladle that had been preheated. From the ladle it was teemed by top-casting into ingots. About 20 pounds of 50 per cent ferrosilicon were placed in the spout before pouring, and 20 to 30 pounds of 80 per cent ferromanganese were added to the charge in the furnace. The weight of metal tapped was 2.1 tons. Some aluminum was added to the tops of the ingot molds.

The electrode consumption is about 20 pounds per ton, at a cost of about 4 cents a pound. Under the poor operating conditions the silica roof lasts about 20 heats. The conducting bottom had been in 60 heats at the time of inspection and was still in good condition.

The furnace force consists of a superintendent, a chemist, a melter, two assistant melters, a ladle man, and one general laborer.

PLANT OF CRUCIBLE STEEL CASTING CO., LANSDOWNE, PA.

At Lansdowne, Pa., the Crucible* Steel Casting Co. has recently built the first Röchling-Rodenhauser steel furnace to be used commercially in the United States. It is used for producing steel for castings from cold scrap.

The current used is single-phase and the furnace is of 2 tons capacity, similar to the furnace shown in figure 27. It is situated 200 feet from the generator, on an operating platform. The lining is a mixture of magnesite and tar and is tamped by hand. The roof is magnesite brick. A motor-driven blower supplies the air for cooling the transformer coils for the furnace.

A 300-kilowatt generator supplies a 25-cycle single-phase current at 480 volts, and is directly driven by a 2-cylinder oil engine. The power cost is estimated at 0.7 cents per kilowatt-hour.

* Von Bauer, C. H., The Röchling-Rodenhauser furnace of the Crucible Steel Casting Co., Lansdowne, Pa.: *Iron Trade Rev.*, vol. 53, 1913, p. 153.

The furnace is charged while hot with cold scrap, some molten steel from the previous run being left in the furnace. At present it is operated only 12 hours a day, making two melts in that period. The general refining process is the same as in other electric steel furnaces. The length of run varies from four to six hours and the power consumption from 800 to 900 kilowatt-hours per ton. Steel has been produced of the following composition—0.30 per cent carbon, 0.30 per cent silicon, 0.49 per cent manganese, 0.03 per cent phosphorus, and 0.03 per cent sulphur.

SUPERREFINING OF MOLTEN STEEL IN THE ELECTRIC FURNACE.

PLANT OF ILLINOIS STEEL CO., SOUTH CHICAGO, ILL.

The 15-ton 3-phase Héroult^a furnace at the South Chicago works of the Illinois Steel Co. until recently was one of the two largest electric furnaces in operation for the manufacture of steel. A wide variety of products was made in an attempt to learn what could be done with the electric furnace. The furnace was primarily intended for the refining of molten Bessemer steel, but some cold scrap has been treated.

DESCRIPTION OF PLANT.

The Héroult furnace at South Chicago has already been described. The steel operating platform of the furnace is about 9 feet above the ground level. Around the furnace on this platform, at convenient points, bins are placed for the materials used in furnace operation. The front part of the furnace platform opens to allow a ladle to be hung in position when steel is poured.

Power for the furnace is supplied by the central plant of the works. Dynamos are driven by reciprocating gas engines, reciprocating steam engines, and high-pressure and low-pressure turbines. Blast-furnace gas is used in the gas engines and under boilers. The power cost is about 0.5 cent per kilowatt-hour. The 25-cycle 3-phase current is stepped down at the furnace from 2,200 volts by three 750-kilowatt transformers. These transformers are arranged so that the number of turns in the primary may be altered to give a secondary voltage of 80, 90, 100, or 110 volts. The usual voltage is 90 volts.

The pouring platform, 30 feet long, enables eight molds to be placed in position for pouring. The furnace is served by a 50-ton crane.

PLANT PRACTICE.

Bessemer pig iron is full blown, in a 15-ton Bessemer converter, in 8 to 12 minutes. The analysis of the Bessemer metal shows 0.05 to

^a Osborne, G. C., The 15-ton Héroult furnace at the South Chicago works of the Illinois Steel Co.: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 205; The South Chicago electric furnace plant of the U. S. Steel Corp.: Met. and Chem. Eng., vol. 8, 1910, p. 179.

0.10 per cent carbon, 0.005 to 0.015 per cent silicon, 0.05 to 0.10 per cent manganese, 0.035 to 0.07 per cent sulphur, and about 0.095 per cent phosphorus. It is then poured directly from the converter to a transfer ladle and drawn to the electric furnace building, about one-fourth of a mile away. As a precaution against the possible formation of a skull in the ladle, the Bessemer charge is blown hotter than in ordinary Bessemer practice.

When the ladle is received at the electric furnace it is lifted by a crane and tilted; the silicious slag is removed by hand rabbling. The metal is then poured into the electric furnace through a spout. The cleaning of the slag and the charging take 5 to 10 minutes. As the metal pours into the furnace the helper shovels iron oxide and lime through the furnace doors to produce a basic oxidizing slag for the removal of phosphorus. In about 30 minutes the furnace is tilted forward and the slag removed in 5 to 10 minutes by hand rabbling. The quantities of carbon, silicon, manganese, and phosphorus have been reduced to a low point in this period.

In the second or deoxidizing stage which now begins sulphur is eliminated as much as possible. The recarburizing agent is added, followed by lime and fluorspar to keep the mass fluid. In about 15 minutes this second slag is fluid, and finely divided coke dust, the deoxidizer, is thrown on top of the slag, forming a reducing atmosphere, as shown by the formation of calcium carbide beneath the electrodes. Thereafter there is a dead melting in a reducing atmosphere. The slag is basic and fluid and contains considerable calcium carbide. It will be seen that the refining practice with molten metal is quite similar to that with cold scrap.

Tests are taken to show the condition of the steel. A small cylindrical test piece is poured and then flattened under a steam hammer at the furnace. If the forged sample appears to be satisfactory, the bath is tapped. The furnace electrodes are raised, and the charge is poured into a ladle, from which it is teemed into ingots.

The quantity of material used in a typical charge is given in the following charge sheet:

Charge sheet of 15-ton Héroult furnace, South Chicago, Ill.

Materials used.	Pounds.	Pounds per tons of 2,240 pounds.
Bessemer blown metal.....	30,000	
Scale.....	700	52.3
Ferromanganese, 80 per cent pure.....	200	14.9
Ferrosilicon, 10 per cent pure.....	60	4.5
Ferrosilicon, 50 per cent pure.....	80	6.0
Recarbonizing agent.....	120	9.7
Fluorspar.....	400	29.9
Coke dust.....	200	14.9
Lime, first slag.....	800	45.0
Lime, second slag.....	600	45.0
Dolomite lining.....	400	29.9
Magnesite lining.....	25	1.8

The time consumed in the different operations is divided as follows:

	a. m.
Previous heat tapped.....	7.00
Metal ordered for.....	7.15
Metal received.....	7.15
Fettling begun.....	7.17
Current on.....	7.27
Removal of slag begun.....	8.00
Removal of slag finished.....	8.11
Tapped.....	8.48

Time of heat, 1 hour 21 minutes.

The temperature of the Bessemer metal poured into the electric furnace is almost $1,500^{\circ}$ C., whereas that of the steel poured from the furnace is about $1,495^{\circ}$ C. At the close of a run the furnace temperature drops from about $1,500^{\circ}$ C. to $1,300^{\circ}$ C. during the 15 minutes necessary to patch the lining for the next run.

On an average 12 heats are made daily. During a series of experiments in which both dephosphorization and desulphurization took place, the average power consumption was 198 kilowatt-hours per long ton. When desulphurization alone was carried on, the average power consumption was 105.9 kilowatt-hours per long ton. The power factor of the furnace varies from 0.80 to 0.90.

The hearth is fettled after each run with dolomite, from 10 to 30 pounds of dolomite being used per ton of steel. The hearth proper consists of 4 parts dead-burned, ground magnesite mixed with 1 part basic open-hearth slag. To this mixture tar enough is added to make the mass plastic for tamping. After being tamped, the furnace is heated with a wood fire for 48 hours; it is then filled with coke, the current turned on, and the bottom fluxed into place. This bottom and the side walls last a long time. A silica-brick roof withstands about a week's use and is repaired or replaced during the regular weekly shutdown. The roof has then been subjected to about 72 heats, although some roofs have stood 129 heats. A new roof is always kept in reserve.

Extensive tests have been made with various kinds of electrodes. The average consumption of either amorphous carbon or graphite electrodes is given as about 6 pounds per ton of steel. As in cold-scrap refining processes, the electrodes are regulated by hand during the first few minutes of a run.

PRODUCTS.

This furnace has produced a greater variety of steel than any electric furnace in operation, the product including high-grade alloy steel, high-grade carbon steel, and ordinary carbon steel. From the ordinary carbon steel produced, rails of a dozen different sections,

billets of all sizes and grades, plates of all sizes and grades, structural shapes, castings, both small and large, and high and low carbon, and forgings of all sizes have been manufactured. Among the alloy steels made are nickel, nickel-chrome, chrome, manganese, and-silicon steels. Some steel has been made from cold scrap.

The main characteristics of the steels produced in the furnace are a comparative freedom from oxidation and from segregation and a higher tensile strength and slightly higher ductility than that of open-hearth or Bessemer steel of the same chemical composition up to 0.4 per cent carbon, where the difference becomes less apparent.

PLANT OF AMERICAN STEEL & WIRE CO., WORCESTER, MASS.

A 15-ton, three-phase Héroult furnace similar to the South Chicago furnace was in operation until recently at Worcester, Mass., for the production of steel for wire from basic open-hearth steel. Owing, presumably, to difficulty in drawing the comparatively high-carbon steel into wire the furnace is not operated regularly at present. The furnace is similar in construction to the furnace at South Chicago. A current of 12,000 volts is received and is transformed to about 110 volts or less at the plant, according to the voltage desired for the furnace. There are three 750-kilowatt transformers, and the furnace is connected to the transformers by bus bars. Electrode regulators similar to those at South Chicago are used. The cost of power is 0.6 to 0.8 cents per kilowatt-hour.

The process is desulphurizing only and hence requires less time than at that at South Chicago. The procedure of one run was as is stated below. The charge of molten basic open-hearth steel contained 0.18 per cent carbon, 0.22 per cent manganese, 0.02 per cent silicon, 0.02 per cent phosphorus, and 0.027 per cent sulphur. As the 12 to 15 ton charge was poured into the electric furnace about 280 pounds of electrode dust were added. Charging took about 5 minutes. When the furnace was charged a mixture made of 700 pounds lime, 350 pounds fluorspar, and 50 pounds of coke was added. In about 30 or 35 minutes 120 pounds of fluorspar were rabbled in. This was followed in 10 minutes by 30 pounds more of fluorspar. About an hour after the charging was completed a sample was taken. One of these contained 0.50 per cent carbon, 0.12 per cent manganese, and 0.019 per cent sulphur. Ferromanganese and pig iron were then added. About 10 minutes after the first sample had been collected another was taken, which contained 0.45 per cent carbon, 0.45 per cent manganese, and 0.02 per cent sulphur. The slag at this time was tested for calcium carbide, then ferrosilicon, sand, and lime were added. The current was then turned off and the charge poured into a ladle and cast into ingots. The finished product contained

0.54 per cent carbon, 0.47 per cent manganese, 0.157 per cent silicon, 0.022 per cent of phosphorus, and 0.015 per cent of sulphur.

The average current strength for a run was 12,310 amperes; the power factor was 0.696; the voltage was 76.2, 97.7, and 73.3 volts; and the power 396, 451, and 404 kw. to the respective phases. The sum of the kilowatts on the three phases was 1,253 kw. and the average length of a run was 1.41 hours, making the total power consumption 1,746 kilowatt-hours. The average power consumption per ton of metal poured is about 152 kilowatt-hours.

The temperature of the steel when charged is about 1,454° C. The temperature of the slag is 1,524° C. The steel is tapped at a lower temperature than it is charged.

The total losses in bus bars, electrodes, and transformers is 6.53 per cent of the current measured at the switchboard. About 2 per cent of this loss is in the transformers. The loss by conduction through the walls is 5.57 per cent of the total power supplied the furnace.

GUTEHOFFNUNGSHÜTTE WORKS, OBERHAUSEN, GERMANY.

An extensive test* has been conducted at Oberhausen, Germany, on the Girod electric furnace (fig. 32) for the refining of steel from molten basic open-hearth metal.

DESCRIPTION OF PLANT.

The Girod furnace installed here is from 2.5 to 3 ton capacity (fig. 22) and is similar to the usual type of Girod furnace.

The single-phase current is supplied by a motor-generator set consisting of a 3,000-volt, 95-ampere, 575-kilowatt, three-phase induction motor driving a single-phase 75-volt, 6,700-ampere, 25-cycle, 500-kilowatt generator. The power factor is 0.8. These machines, as well as all conductors carrying high-tension currents, are placed in a room adjoining the furnace. The loss in transforming from three-phase to single-phase current is about 14 per cent. On the furnace switchboard there are ammeters and voltmeters in the low-tension circuit, with a wattmeter in the primary circuit. The excitation-current regulator for the furnace voltage, the manual and automatic regulator for the electrodes, and the device for tilting the furnace are all controlled from the switchboard.

REFINING PRACTICE.

In continuous operation it is possible at this plant to run 8 heats in 24 hours, a total of 25 tons of steel. The basic open-hearth furnaces

* Mueller, A., The manufacture of steel in the Girod electric furnace: Metall. Chem. Eng., vol. 9, 1911, p. 581.

Figure 33 shows diagrammatically the chemical reactions during a heat in the Girod furnace and illustrates the changes of the carbon, manganese, phosphorus, sulphur, and silicon contents.

The results of the analyses of 19 samples are here plotted. Sample 1 is that of the charge, sample 2 was taken after addition of 15 kg. (33 pounds) of ore, sample 3 after addition of 50 kg. (110 pounds) of ore. Samples 4 and 5 were also taken during the oxidation period, sample 5 being taken just before removal of the oxidation slag. Samples 6 to 19 were taken during the deoxidation period. Sample 6 was taken after the addition of 16 kg. (35 pounds) of Girod carbons and 5 kg. (11 pounds) of ferromanganese; sample 7 after the addition of 12 kg. (26.5 pounds) of ferrosilicon (50 per cent Si) and 40 kg.

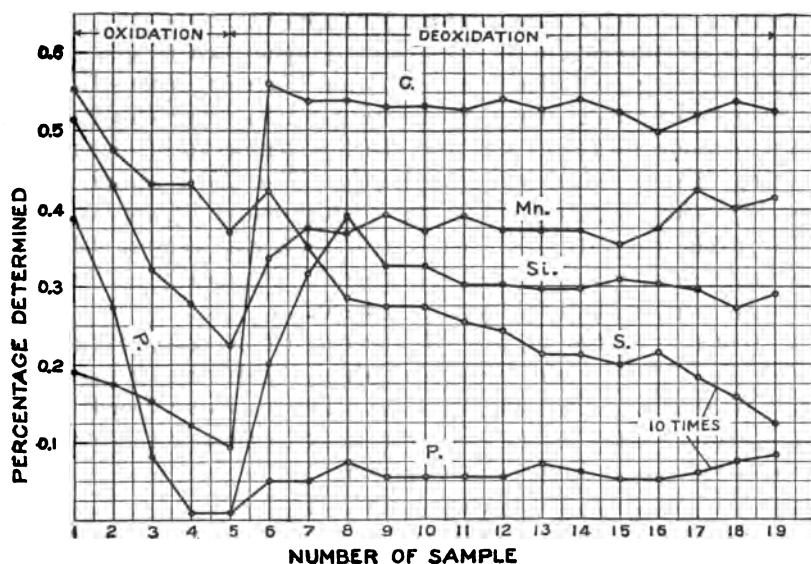


FIGURE 33.—Results of analyses of 19 samples of metal during one heat.

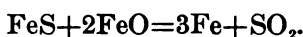
(88 pounds) of refining slag; sample 8 after the addition of 4 kg. (8.8 pounds) of ferrosilicon; sample 9 after the addition of 20 kg. (44 pounds) of refining slag; sample 10 after the addition of 3 kg. (6.6 pounds) of ferrosilicon; sample 11 after addition of 2 kg. (4.4 pounds) of powdered petroleum coke; sample 12 after the addition of 1 kg. (2.2 pounds) of ferromanganese (80 per cent Mn); sample 13 after the addition of 8 kg. (17.6 pounds) of lime; nothing was added when samples 14 and 15 were taken; sample 16 was taken after the addition of 10 kg. (22 pounds) of lime and fluorspar and 2 kg. (4.4 pounds) of powdered petroleum coke; and sample 17 after addition of 2 kg. (4.4 pounds) of ferromanganese. Nothing was added after that, sample 19 being the last.

The oxidation period must be preceded in many cases by a melting period of a length corresponding to the interval between charges. A longer interval causes the steel to solidify in parts. During this time ore is added gradually to the bath in the furnace.

The carbon oxidizes slowly in spite of the high temperature in the furnace. The curves shown in figure 33 are based on data obtained from a heat in which 0.10 per cent carbon was oxidized in 55 minutes, the amount oxidized being equivalent to 53 per cent of the original carbon content. Although the manganese was present in a more concentrated state than the carbon, only 58 per cent of the original manganese content was oxidized in the same time. The amount oxidized is equal to a decrease in manganese from 0.52 to 0.22 per cent.

Phosphorus can be more completely removed than the other impurities since the proportion of the oxidizing agent can be governed at will and the slag can be kept very basic, and these conditions facilitate the removal of phosphorus. As decarburization progresses the phosphorus content decreases steadily, the low temperature of the bath favoring the oxidation of phosphorus. The third sample taken (fig. 33), after a charge of 50 kg. (110 pounds) of ore had been added, contained only 0.008 per cent phosphorus, against 0.039 per cent in the first, a reduction of 80 per cent. Practically all the phosphorus is removed before slagging proceeds.

As shown in figure 33, the sulphur content decreases from 0.055 per cent to 0.037 per cent during oxidation, the decrease being 33 per cent. Wüst claims that slag containing ferrous oxide dissolves iron sulphide to a certain extent. In stronger concentration it reacts with ferrous oxide, according to the equation:



The sulphur dioxide escapes and can be clearly recognized. But a part of the sulphur may still remain dissolved in the slag together with free ferrous oxide, as is shown by the sulphur in the oxidation slags. The average decrease in sulphur content during the oxidation period is 26 per cent.

The oxidation slag is drawn off, and the last trace of phosphorus is removed from the bath by mixing lime with it. This method avoids rephosphorization of the bath by the carbon and silicon additions in the following period. One slag is sufficient to remove all the phosphorus except traces.

The deoxidizing slag is produced by adding to the bath about 20 kg. (44 pounds) of lime, 3 kg. (6.6 pounds) of sand, and a like quantity of fluorspar, and also 1 kg. (2.2 pounds) of ferrosilicon (50 per cent silicon) per ton of steel. This slag dissolves the ferrous oxide that has been forming on the unprotected surface of the metal and thereby becomes black,

The reduction of the ferrous oxide in the slag is indispensable for the deoxidation and desulphurization of the bath. For this deoxidation the silicon from the added ferrosilicon is not sufficient, and 1 to 2 kg. (2 to 4 pounds) of powdered petroleum coke are added to insure complete deoxidation. The slag obtained disintegrates in the air quickly to a white powder, indicating that the ferrous oxide has been reduced to traces.

With high-carbon products, petroleum coke or waste pieces of carbon electrode are added in proper proportion to the bath as required for purposes of carburization prior to the formation of the deoxidation slag. As soon as the slag is readily fluid and free from oxide, ferromanganese, ferrosilicon, and other alloys are added.

The addition of these alloys which are often impure may increase the phosphorus and sulphur content considerably. Thus, in figure 33 sample 6 shows that the phosphorus has increased from traces to 0.08 per cent, owing to the addition of ferromanganese with a phosphorus content of 0.4 per cent. The chemical composition of the slag at different periods of the melt follows:

Composition of slags.

Constituent.	Oxidizing slag.	Deoxidizing slag.		Finishing slag.
		After carburization.	After main charge.	
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
CaO.....	41.69	59.50	69.45	75.85
SiO ₂	9.58	26.78	17.90	13.20
FeO.....	28.36	1.02	.52	.13
MnO.....	.99	1.44	.52	Trace.
S.....	.62	.39	.90	1.22
Fe ₂ O ₃	6.43	.23		
Al ₂ O ₃	1.73	1.67	1.27	1.73
MgO.....	6.99	5.54	5.78	4.22
P ₂ O ₅	1.47	.07	.12	.09
Oxide of bases.....	.20	.20	.22	.22
Oxide of acids.....	9	15	10	8
Appearance of slag.....	Black and streaky.	Grayish brown sand.	White sand.	White powder.

The sulphur is removed in three ways: During oxidation, through the formation of sulphur dioxide by the action of slag rich in ferrous oxide; in the deoxidation period, through the action of the white slag, free from ferrous oxide; and through the union of silicon with sulphur, forming silicon sulphide.

The quiet melting of a cold scrap-steel charge has been recognized as a great advantage of the Girod furnace. Figure 34 shows the fluctuations of the kilowatt consumption as traced by a recording wattmeter, starting with a liquid charge. The regulation of voltage and current is almost always automatic; hand regulation, with the auto-

matic regulator disconnected, is used only during the beginning of a run. In figure 34 the letters represent the time of the operations as given: A, tapping; B, C, and D, additions of flux; E, carburization; F, slagging; G, slag readily fluid; H, bath boiling; I, additions of ore; J, slag not fluid; K, charging.

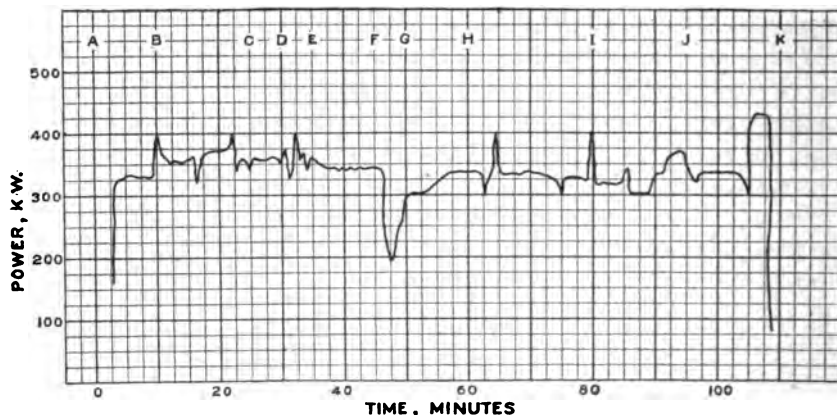


FIGURE 34.—Power fluctuations of 2.5-ton, single-phase, Girod steel furnace during a heat.

The size of the furnace, the duration of the intervals between successive heats, the cooling of the furnace during these intervals, the condition of the charge, the quality of steel desired, and other factors all affect the energy consumption decidedly.

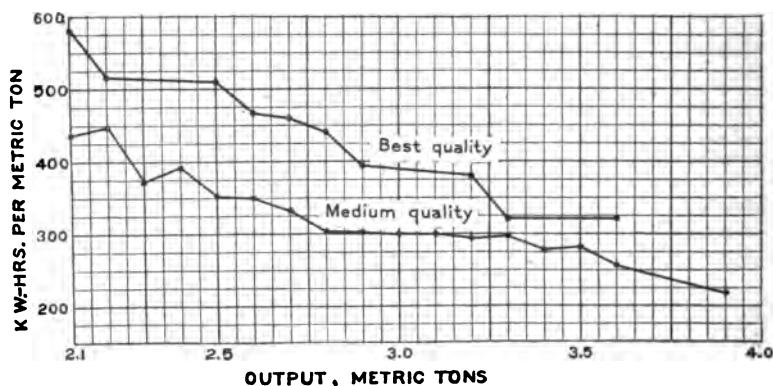


FIGURE 35.—Specific-energy consumption as function of weight of charge, Girod furnace.

Figure 35 shows the influence of the weight of charge on the specific-energy consumption per ton. The results are averages taken from 500 successive heats, including some of alloy steels. The energy consumption depends a great deal on the weight of the charge and decreases in large furnaces.

The loss of heat through radiation and conduction and the fluctuations of temperature decrease with increasing size of furnaces, the temperature being much more easily regulated in large furnaces than in small ones.

The minimum economical weight of charge is about 2,800 kg. (fig. 35), the specific-energy consumption not changing materially with increasing weight up to 3,300 kg.

The specific-energy consumption depends still more on the temperature of the furnace, and this depends greatly on the duration of the intervals between charges. The fall of temperature after tapping is rapid compared with that of an open-hearth furnace, the temperature of which decreases to only 1,000° C. in 25 hours and decreases after-

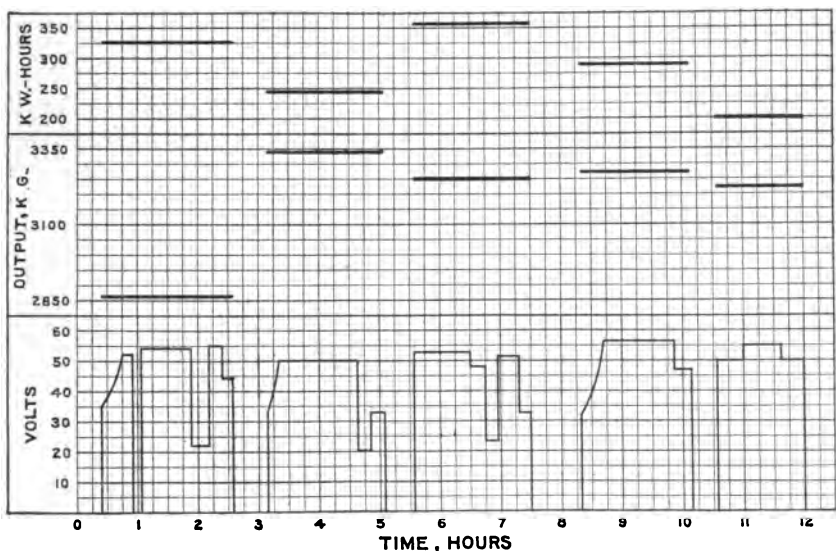


FIGURE 36.—Voltage, output, and power consumption for 2.5-ton, single-phase, Girod steel furnace.

wards at the rate of 20° C. per hour. Under such conditions it is possible to get a high thermal efficiency with the Girod furnace only by charging it, at the latest, 20 minutes after the last tapping. The temperature of the interior of the furnace is then still about 1,300° C. The energy consumption per ton is then 188 kilowatt-hours for medium-grade steels and 270 kilowatt-hours for high-grade steels (fig. 36).

Under favorable conditions it was possible in a Girod 3-ton furnace to refine 3,654 kg. (8,039 pounds) of steel with an energy consumption of 158 kilowatt-hours per ton, including refining by oxidation, deoxidation, and desulphurization. Large Girod furnaces give more favorable results, as they can be charged immediately after tapping, while they are at a temperature of 200° C. higher.

The silica brick roof of the furnace lasts for 60 to 70 heats. The walls are attacked most at the slag line. This is fettled between charges. After 120 heats it is necessary to reline the walls. The hearth bottom is also repaired then, as by that time it has sunk about 2 inches in the center. The hearth does not have to be pulled out when it is repaired.

The heat loss due to cooling of the bottom electrodes is 1.01 per cent of the energy consumption, and in producing 3.5 tons of steel the cooling of these electrodes consumes about 2.9 kilowatt-hours per ton of steel poured. The heat lost in cooling the silica roof around the carbon electrode is 3.65 per cent of the total energy supplied. With a production of 3.5 tons, 10.5 kilowatt-hours per ton would be lost in cooling the upper electrode.

PRODUCTS.

The analyses, tests, and uses for some of the products of this Girod furnace follow. Similar products can be made in any commercial electric steel furnace.

Analyses, tensile strength, and uses of various electric steels.

Va.	W.	Cr.	Ni.	C.	Mn.	P.	S.	Si.	Tensile strength.			Use.
									Weight per square millimeter.	Elongation on 200-millimeter bar.	Reduction of cross section.	
P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	Kg.	P. ct.	P. ct.	
.....				0.13	0.54	Tr.	0.014	0.16	41.2	30	65.6	Angle.
.....				.13	.70	0.018	.020	.08	43	31	62.3	Do.
.....				.19	.56	Tr.	.010	.13	47	31	55	Do.
.....				.22	.90	.010	.017	.25	58	27.5	52.8	Iron.
.....				.25	.88	Tr.	.017	.26	60	23.5	49.1	I-beam.
.....				.33	.74	.010	.010	.20	65	19	42	Engine and machine parts.
.....				.64	.62	.010	Tr.	.47	98	13.2	23.5	Square blocks.
.....				.53	.68	.008	.010	.37	86.9	12.8	30.3	Flat steel bars.
.....				.57	.70	Tr.	.008	.40	89.4	10.7	21.4	Do.
.....				.72	.42	.020	.010	.34	97.1	6	23.8	Dies.
.....				.82	.48	.006	.010	.25	99.5	5.8	20.1	Punches.
.....				.70	.72	.010	.010	.36	83.8	11.5	19.8	Tramway car-wheel tires.
.....				.47	.78	.010	.012	.24	71.5	17.5	32.9	Railway car-wheel tires.
.....				.36	.78	.013	.010	.17	64	19	40.7	Axles.
.....				.48	.78	.007	.019	.28	74.4	13.3	24.9	Locomotive wheel-tires.
.....				.52	.78	.015	.018	.19	78.7	11.1	20.5	Do.
.....				.48	.84	.007	.018	.23	72	17.7	44	Shafting.
.....									72	16.5	45.2	Do.
.....				1.89	.26	.78	Tr.	.12	61.8	23.5	48.6	Spindles.
.....				1.98	.20	.72	.012	.13	54	22	58.6	Do.
.....				1.04	1.02	.58	Tr.	.12				Knives, shears, chisels.
.....				.49	.82	.012	Tr.	.32	79.7	15	40.4	Tools.
0.19	1.50	.92		.46	.60	Tr.	.012	.22	79.6	10.2	32.9	Pieces for forging.
.....		.85							82.1	33	59.5	Quenched at 700° C. in oil.
.....									59.6	36	63.8	Quenched at 700° C. in water.
.....				.68	.32	.010	.010	.23	93.3	9	18.8	Rolls for making grooved rail.

WORKS AT VOLKLINGEN, GERMANY.

Two Röchling-Rodenhauser furnaces at Volklingen, Germany, are in operation for the refining of molten basic Bessemer steel, with the production of steel varying in grade from rail steel to high-priced tool steel. At this plant the Röchling-Rodenhauser furnace was developed from the original Kjellin design.

One of the furnaces is of 8 to 10 tons capacity (fig. 27), taking 600 kw., supplied by a 25-cycle single-phase current at a voltage of 4,000 to 5,000 volts. The 2 to 3 ton three-phase furnace requires 200 to 250 kw. at a pressure of 400 volts on a 50-cycle circuit. The details of these furnaces have been previously described.

METHOD OF RELINING FURNACES.

In relining the furnaces the furnace castings are first protected by layers of fire brick, which serve as the principal protection against radiation losses and rarely require replacement. On the fire brick a layer of a mixture of calcined dolomite and tar is rammed to the level of the hearth bottom. Cast iron templates are then set and dolomite mixture is rammed behind them on the sides. After the templates are removed, the walls are finished with dolomite brick.

The lining is dried by placing cast-iron rings in the channels and heating the rings by the passage of current. When the tar in the mixture is burned out the roof is put on and molten pig iron is charged to complete the sintering of the hearth.

The time required for relining a 3-ton furnace is as follows: Removing old lining and ramming in new, 24 hours; burning out tar with hot rings, 6 hours; completing the sintering of the lining, 6 hours—total, 36 hours. During burning and sintering about 230 kilowatt-hours are consumed. Such a lining stands on the average 55 charges, or 360 to 500 tons of hot metal, according to the steel produced.

REFINING PRACTICE.

The molten metal charged into the furnaces at Volklingen from the basic Bessemer converters contains 0.1 per cent carbon, 0.5 per cent manganese, 0.08 per cent sulphur, and 0.08 per cent phosphorus. At times molten basic open-hearth steel that has been refined, containing about 1.22 per cent carbon, 0.38 per cent manganese, and 0.209 per cent silicon, is put in the electric furnace to allow removal of gases, to adjust the carbon content and to add alloys.

Dephosphorizing and desulphurizing are carried on as in the arc furnace, but not at so high a temperature, although the temperature is sufficient for the purpose, and is more evenly distributed through the charge. Also most of the desulphurizing is caused by the action of ferrosilicon and lime on ferrous sulphide, as the temperature is not

high enough to permit the formation of calcium carbide. Hence more ferrosilicon is charged in the induction furnace than in the arc furnace.

The furnace is started cold, rings of steel are placed in the channels, and molten metal is poured on them. The voltage of the secondary current, through the pole plates, is higher than usual at the start in order to warm the hearth more quickly and make it conducting. Formerly it was customary to pass current through the furnace on Sunday to keep it hot, but now the doors are closed and the current is shut off entirely. After the furnace has stood 30 hours part of the current is turned on and the amount is gradually increased until in 4 to 6 hours the furnace is at working temperature. After each run the furnace is practically emptied. In using cold scrap about two-thirds of the charge is molten steel and the rest is cold scrap.

With hot metal the length of a heat varies from 1 hour 15 minutes to 3 hours. With a cold charge the time required is 3 to 6 hours. To produce a rail steel containing 0.5 per cent carbon, 0.8 per cent manganese, 0.04 per cent sulphur, and 0.05 per cent phosphorus, about 100 to 125 kilowatt-hours per ton are necessary with a hot charge of Bessemer metal. For high-grade steel containing 0.05 per cent carbon, 0.25 per cent manganese, and trace of sulphur and phosphorus, the power consumption is about 300 kilowatt-hours per ton. The power factor averages about 0.60 with both the single and the 3-phase furnaces. With cold scrap the power consumption is claimed to be 580 kilowatt-hours per ton. The power cost at Volklingen is about 1.5 cents per kilowatt-hour.

PRODUCTS.

Steel of many grades has been made at Volklingen in the induction furnace, including high-grade tool, alloy, and rail steels. Some of the mild-steel products and the power consumption are as follows:

Mild-steel products of Röchling-Rodenhauser furnace.

	C.	Si.	Mn.	P.	S.	Time.		Power consumption per ton.	Total energy consumed.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Hours.</i>	<i>Minutes.</i>	<i>Kw.-hrs.</i>	<i>Kw.</i>
Charged.....		0.028	0.534	0.037	0.081	1			
Poured.....		.022	.307	.005	.057		50	250	628
Charged.....		.034	.478	.060	.077	2			
Poured.....		.018	.309	.011	.053		25	274	685
Charged.....	0.128	.032	.590	.064	.085	1			
Poured.....		.140	.618	.020	.065		45	235	587
Charged.....	.116	.018	.679	.054	.097	2			
Poured.....		.122	.695	.028	.098		5	282	705

PLANT OF LA GALLAIS METZ & CO., DOMMELDINGEN, LUXEMBURG.

At Dommeldingen, Luxemburg, is the largest installation of Röchling-Rodenhauser furnaces in operation. There are three single-phase $3\frac{1}{2}$ -ton furnaces, similar to the one shown in figure 27, and one three-phase 2.5-ton furnace. The single-phase furnaces take 350 to 400 kw. at 3,500 volts, 50 cycles. The three-phase furnace requires 275 kw. at 500 volts, 50 cycles. The power factor of the single-phase furnace averages about 0.6, and that of the three-phase furnace is sometimes as high as 0.7.

Pig iron is made into steel in a gas-heated 20-ton tilting furnace. When 3.5 tons of the steel is transferred to an electric furnace for further refining it is replaced by 3.5 tons of pig iron. The gas furnace acts as an oxidizing mixer. The steel in the gas furnace has 0.26 per cent carbon, 0.082 per cent phosphorus, and 0.03 per cent sulphur. For ordinary castings the steel is in the electric furnace about 2 hours, for the best machinery castings 3 to $3\frac{1}{2}$ hours.

The three-phase furnace was not in use at time of inspection, as faulty design had resulted in poor operation.

Some of the results obtained in practice at Dommeldingen were as follows:

Results obtained from furnace practice at Dommeldingen.

Sample.	Chromium content.	C, Si, Mn, P, and S content.				
		Carbon.	Silicon.	Manganese.	Phosphorus.	Sulphur.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Pig iron.....		3.50	0.60	1.20	1.80	0.12
Mixture from gas furnace.....		1.62		.66	.46	.035
Tappings from gas furnace.....		.26		.30	.08	.030
Steel from electric furnace.....		.35				
Castings.....		.40	.25	.50	.020	.010
Hard fine steel.....		.85	.23	.26	.011	.009
Chrome steel.....	0.87	1.17	.19	.16	.014	.009

CHARACTERISTICS OF OPERATION OF ELECTRIC STEEL FURNACES.

DESIGN.

From what has been said it is clear that electric steel furnaces vary widely in design, but they may be divided into two general classes, arc furnaces and induction furnaces. The general features of the operation of the furnaces in a class are similar. Steel of about the same grade can be made in any of the commercial electric furnaces, but from their design some furnaces are better adapted to one purpose than to another. In general the power consumption and efficiencies of the various types are about the same. Hence the choice of the type of furnace for a specific purpose should be determined by the factors of the individual case.

Several general points should be considered in choosing the type of furnace. In comparing the arc and the induction furnace it should be remembered that the arc furnace is better adapted to melting cold scrap than is the induction furnace, because the latter must be charged with molten metal at the start to insure quick and regular heats. The power factor of the induction furnace is lower than that of the arc furnace and is much lower than that of most public power service lines. Hence, an induction furnace that might be a satisfactory load on a private plant carrying no other load, on a public line might lower the power factor of the whole system to an objectionable degree. If an induction furnace is built where power must be obtained from a public service line, the company operating the furnace may be required to put in a motor-generator set at its own expense. In general, the induction furnace makes a more steady demand on the central power plant than the arc furnace, because there is no constant breaking of the arc which may cause fluctuations.

In choosing an arc furnace for refining cold scrap it seems that the load on the power line is more satisfactory, if the furnace is of the conducting-hearth type with several upper electrodes in parallel rather than of the type with two arcs in series with the bath. This is because the arc breaks less frequently in the former than in the latter. For continuous steady operation on either cold scrap or molten steel, one furnace is probably about as good as the other as regards mechanical features, power consumption, and heat losses, although some consider the water-cooled electrodes of the conducting hearth and broken hearth to be objectionable. An arc furnace in which heating is by radiation of the arc has a higher power consumption than the other types. Electrode consumption and repair cost are also higher.

POWER CONSUMPTION.

The power consumption of furnaces depends so much upon the materials being treated, the process of operation, and the product desired that no satisfactory comparisons can be made. Also, the point in the circuit at which the reading is taken for figuring the basis of power consumption may be so variously selected that one furnace may show a much lower power consumption than the other when it should show about the same. With cold scrap the power consumption of the various arc furnaces varies from 459 to 1,200 kilowatt-hours per ton. Aside from the points mentioned, this wide variety of results may be due to the amount of refining done. Furnaces in which only one refining slag is used take about 600 kilowatt-hours per ton on an average, while if two slags are used 800 to 1,000 kilowatt-hours per ton are necessary. Power consumption per ton of metal is also influenced by the size of the furnace. For example, a 10 to 12 ton Girod

furnace uses about 12 per cent less power per ton than a 2.5 to 3 ton furnace of the same type.

In refining molten steel in the electric furnace, the power consumption has varied from 100 to 300 kilowatt-hours per ton. Rail steel of this class takes about 100 kilowatt-hours per ton when one slag only is necessary and about 200 kilowatt-hours per ton with two slags in a 15-ton furnace. Tool steel is made from molten metal in a 2 to 3 ton furnace with a power consumption of 150 to 300 kilowatt-hours per ton.

POWER FACTOR.

The power factor is the ratio of the number of effective watts in an alternating-current circuit to the number of volts multiplied by the number of amperes. Any self-induction in the circuit causes a difference in phase between the electromotive force or voltage and the current or amperage, and the current will thus lead or lag behind the voltage; that is, the two do not reach their maxima at the same moment. An amount of energy equal to the number of amperes multiplied by the number of volts is surging through the circuit, but, as the electromotive force and current do not reach their maxima together, the product does not represent the effective energy that is usefully consumed at the receiving end.

The generators have to supply the total wattage and the line has to conduct it. Hence to offset a low power factor large generators and thick conductors must be used. For example, a power factor of 0.5, or 50 per cent, means that conductors of double size must be used to get the same number of effective watts as with a power factor of 1, and it also means that generators of double capacity must be used to produce the same useful output. In such a case there would still be objection raised by the power company, for the voltage regulation of the line and generators would be poor, and the low power factor would act as a brake on the circuit, throwing back on it all power not effectively used and causing the generating machinery to heat. If the company using a furnace with a low power factor generates its own power, it must consider the increased capital investment on account of larger generating capacity.

If the ordinary loads of a central power station are considered, an operating power factor above 0.95 will be obtained only when practically all of the load consists of synchronous motors or rotary converters that may be operated with a power factor of about 1. The power factor for a plant having a large proportion of induction motors is about 0.70. Hence in operating several electric furnaces the necessity of keeping the power factor of the single unit as much above 0.80 as possible is evident or the power factor of the whole plant may drop below 0.70.

Without discussing the causes in furnace design that influence the power factor, it may be said that there are three ways of reducing the trouble. One is to reduce the frequency of the alternating current, but, although this should be considered when a new plant is to be erected, in an established works such a change means new electrical machinery of special design and higher cost. A more practical way is to increase the resistance of the furnace, which results in a higher working voltage and a higher power factor. The third method is to decrease the quantity of iron surrounding the brickwork of the furnace as much as possible, as this iron causes the formation of lines of force that tend to lower the power factor. Tie-rods and buck staves may be made of nonmagnetic special steel, or a strip of copper may be inserted to break the continuity of the iron surrounding the circuit.

The effect of frequency has been mentioned. It is for this reason that most electric steel furnaces are operated with a current having a frequency of 25 cycles rather than with the 60-cycle current ordinarily used in commercial practice. Up to the time of the design of the combination induction furnace the great drawback to the induction furnace as originally built was the necessity of using a current with a frequency as low as 5 to 15 cycles in order to obtain a power factor over 0.5. With the combination induction furnace a power factor of 0.6 can be obtained with a frequency of 25 cycles.

The power factor of a furnace decreases if the size is increased and the voltage and frequency remain the same. For instance, the power factor of a 25-ton single-phase Héroult furnace varies from 0.85 to 0.95 with a current frequency of 25 cycles and a voltage of 110 volts, whereas in a 15-ton three-phase furnace the power factor is 0.70 to 0.80 with a current frequency of 25 cycles and a voltage of 90 volts.

The power factor of the combination induction furnace varies from 0.60 in an 8 to 10 ton furnace, single-phase, frequency 25 cycles, to 0.70 in the 3-ton furnace, three-phase, frequency either 25 or 50 cycles. The power factor of the 1.5-ton induction furnace is about 0.5, using a current with a frequency of 15 cycles. The power factor of induction furnaces, owing to the large proportion of iron used, decreases rapidly with increase in size, so that it would seem that this might prevent an increase in size over 8 to 12 tons.

In general the arc furnace with the conducting hearth tends to have a slightly lower power factor than the arc furnace with the upper electrodes in series and a nonconducting hearth. The power factor of all designs of commercial electric steel furnaces of the arc type is above 0.70 for the sizes over 5 tons, and above 0.80 for the smaller sizes.

ELECTRODES.

With improved methods of making amorphous-carbon electrodes the electrode consumption of electric steel furnaces has been considerably reduced. The manufacture of electrodes with threads for continuous feeding has reduced the stumpage loss, which was formerly as much as 50 per cent in some cases, and the electrodes are now much more uniform in character. Threaded amorphous carbon electrodes of foreign manufacture sell for about 2 to 3 cents per pound f. o. b. at the maker's plant. Domestic electrodes sell at about the same price but are not made threaded for continuous feeding. The use of graphite electrodes for steel furnaces is not economical because of the high cost per pound and because the consumption per ton of steel produced is almost as large as that of amorphous-carbon electrodes. The manufacturers of graphite electrodes claim that the gain in electrical conductivity offsets the higher cost, although the losses by heat conductivity would be greater unless the graphite electrodes were carefully designed.

For melting and refining cold scrap the consumption of amorphous-carbon electrodes varies from 15 to 50 pounds per ton. The consumption seems to be considerably higher in furnaces of the nonconducting-hearth type with two electrodes than in the conducting-hearth type with one electrode. The difference is probably due to the additional electrode, for in conducting-hearth furnaces of larger size having two to four electrodes the consumption per ton is greater than in the single-electrode furnace. In furnaces used for refining molten steel the wear on the electrodes is not as great and the consumption is much lower, being 6 to 12 pounds per ton.

THE LINING.

The rate of wear of the lining varies so much that it is almost impossible to state general figures. The furnace bottom will last 200 to 400 heats on cold charges and indefinitely on hot metal. The sides are repaired about every 100 heats but are not completely torn out. In most furnaces the hearth has a dolomite bottom and magnesite brick on the sides. The roof is generally silica brick, and, in an arc furnace, lasts 40 to 120 heats. In the arc furnaces operating under the best conditions the roof lasts about 70 heats. In furnaces with magnesite hearth, sides, and roof the lining cost per ton of metal is high. In some of the more recent furnaces the lining on the sides is about half as thick as that formerly used. The thinner lining increases the capacity and does not seem to affect the conduction losses enough to increase decidedly the power cost per ton of output. The hearth lining of an induction furnace lasts about 55 heats and the roof an indefinite period.

ESSENTIAL FEATURES OF REFINING IN AN ELECTRIC FURNACE.

The general process of refining steel in the electric furnace, as has been stated, comprises oxidation, followed by reduction or deoxidation. Owing to the nature of the heat supply, it is possible to have the atmosphere oxidizing, reducing, or neutral. This is not possible in the open-hearth furnace owing to the gases introduced for heating.

The function of slags* in refining steel in electric furnaces is somewhat different from that in refining in open-hearth furnaces. First, all oxygen introduced into the electric furnace must be in the solid form, whereas the open hearth has an unlimited supply in the air admitted. Oxidation can be conducted in an ordinary open-hearth furnace as in an electric furnace, except that the higher temperature of the latter hastens the melt. Deoxidation in which carbon is the chief agent can be performed only in the electric furnace.

During the oxidizing period sulphur is removed only to a small degree in the open-hearth furnace but much more efficiently in the electric furnace, especially when manganese ore is used. It is likely that part of the sulphur forms sulphur dioxide and passes off with the gases. The phosphorus is eliminated in the oxidizing period, just as in the open-hearth furnace.

Sulphur is removed in four ways: (1) As calcium sulphide, formed by the action of lime and carbon on ferrous sulphide at the high temperature of the arc furnace; (2) as calcium sulphide from the reaction of lime, ferrous sulphide, and calcium carbide at the higher temperatures of the arc furnace; (3) as calcium sulphide through the reaction of lime, ferrous sulphide, and silicon at the lower slag temperatures of the induction furnace; and (4) as iron sulphide from the action of ferrous oxide on calcium sulphide. The chemical reactions are as follows:

- (1) $\text{FeS} + \text{CaO} + \text{C} = \text{Fe} + \text{CaS} + \text{CO}.$
- (2) $3 \text{FeS} + 2 \text{CaO} + \text{CaC}_2 = 3 \text{Fe} + 3 \text{CaS} + 2 \text{CO}.$
- (3) $2 \text{FeS} + 2 \text{CaO} + \text{Si} = 2 \text{Fe} + 2 \text{CaS} + \text{SiO}_2.$
- (4) $\text{CaS} + \text{FeO} = \text{FeS} + \text{CaO}.$

The final reaction is reversible, acting from left to right in an atmosphere even slightly oxidizing, whereas the action is from right to left in the first three reactions.

From the equations it may be seen that in both the induction and the arc furnace the formation of a highly basic slag is essential, but desulphurization in the former, owing to its lower temperature, is brought about chiefly by the reducing action of silicon upon the oxides, whereas in the latter the reducing agent is carbon at a higher temperature.

*Ambery, R., *Slags in electric steel refining*: Metall. Chem. Eng., vol. 10, 1912, p. 601.

In the arc furnace complete deoxidation is recognized by the presence of calcium carbide in the slag. When deoxidation of the bath is complete the contents of the electric furnace represent the nearest approach to ideal equilibrium between different chemical compounds that has ever been accomplished in large metallurgical operations. In the converter and the open hearth the metals are under the action of air and gas, in the crucible the metal takes up carbon and silicon, whereas in the electric furnace the action of the metal on the basic lining is very slight and there is no exchange of elements between metal and slag, except that if the dephosphorizing slag of the oxidizing period has not been completely removed before deoxidization begins some of the phosphorus in this slag and some in the new slag will be reduced. This last point must be carefully watched in all refining in the electric furnace.

Although the induction furnace can not attain the high temperature necessary for desulphurizing by use of calcium carbide as can arc furnaces, the results obtained in refining practice are about equal.

COST OF PRODUCING STEEL IN THE ELECTRIC FURNACE.

The cost of making steel in the electric furnace varies with local conditions. The cost of power does not enter so largely into the final cost as it does in some other electrometallurgical processes, especially the refining of molten steel. Plants are operating successfully under a power cost of 1 cent per kilowatt-hour in localities where material can be obtained at the price common to other processes. Plants such as the one at Ugine, France, have been established in remote localities, where the cost of power is very low, 0.2 cent per kilowatt-hour, but the cost of material is high.

COST OF PRODUCING STEEL IN THE GIROD FURNACE.

Girod^a estimated in 1912 the cost of producing steel from cold scrap in the Girod furnace at Ugine as follows:

Cost of refining steel from cold scrap in Girod furnace at Ugine.

Item.	2.5 to 3 ton furnace.	10 to 12 ton furnace.
Raw materials:		
Scrap, 1,100 kg., at \$15 per ton.....	\$16.50	\$16.50
Slag.....	.46	.46
Deoxidizing and recarburizing additions.....	.70	.70
Producing cost:	\$17.66	\$17.66
Electric power, 850 and 750 kilowatt-hours, at 0.2 cent per kilowatt-hour.....	3.40	3.00
Electrodes, at \$64 per ton.....	.60	.70
Wages.....	.60	.30
Maintenance and repairs.....	2.40	1.60
	7.00	5.60
Total cost per ton of steel.....	24.66	23.26

^a Girod, P., The electric steel furnace in foundry practice: Metall, Chem, Eng., vol 10, 1912, p. 663.

Girod estimated the cost of producing steel from molten steel at the same plant as follows:

Cost of refining steel from molten charge in Girod furnace at Ugine.

Item.	2.5 to 3 ton furnace.	10 to 12 ton furnace.
Raw materials:		
Liquid steel, 4 per cent loss in heating, 1,040 kg., at \$16 per ton.....	\$16.64	\$16.64
Slags.....	.40	.40
Deoxidizing additions.....	.70	.70
Producing cost:	\$17.75	\$17.75
Electric power, 275 and 300 kilowatt-hours, at 0.2 cent per kilowatt-hour.....	1.10	.80
Electrodes, at \$64 per ton.....	.25	.25
Wages, 8 heats per 24 hours.....	.20	.20
Maintenance and repairs.....	.80	.50
	2.35	1.75
Total cost per ton of steel.....	20.09	19.49

The estimates of Girod do not include expense for ingots, molds, superintendence, laboratory, amortization, general charges, or royalty charge. These figures apply only to conditions such as prevail at Ugine, where power is very cheap and the cost of material is high.

COST OF PRODUCING STEEL IN THE GRÖNWALL FURNACE, SHEFFIELD, ENGLAND.

The Electro-Metals Co., owners of the Grönwall patents, estimated the cost of production in 1912 for a 2-ton Grönwall furnace operating at Sheffield, England, as follows:

Cost of refining steel from molten charge, Grönwall furnace.

Item.	Without dephosphorizing.		With dephosphorizing.	
Number of charges per week.....	40		30	
Tons per week.....	80		60	
Cost per ton of steel:				
Power at 0.5 cent per kilowatt-hour.....		\$1.00		\$1.25
Repair of roof.....		.16		.20
Dolomite (100 pounds per charge).....		.17		.17
Electrodes, at 4 cents per pound.....		.18		.22
Sundries.....		.14		.16
Royalty.....		1.50		1.50
		3.15		3.50

The items of labor, amortization, interest, raw materials, and general charges are omitted from the estimate given above. The figures refer to the production of steel with a sulphur and a phosphorus content below 0.02 per cent.

COST OF PRODUCING STEEL IN RÖCHLING-RODENHAUSER FURNACES.

Kjellin* in 1909 estimated the cost of production of high-grade steel for castings at Volklingen, Germany. The cost under German

* Kjellin, F. A., Induction and combination furnaces: Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 172.

conditions with a 7-ton, three-phase Röchling-Rodenhauser furnace, using fluid basic Bessemer steel made at Volklingen, is estimated by Kjellin* as follows:

Cost of producing steel for castings in a 3-phase Röchling-Rodenhauser furnace, from molten metal.

	Cost per ton of steel.
Lining:	
Raw materials.....	\$0.304
Wages.....	.043
Heating up furnace:	
2,000 kilowatt-hours at 2.17 cents for 160 tons.....	.272
Refining:	
300 kilowatt-hours per ton at 2.17 cents per kilowatt-hour.....	6.52
Wages.....	.425
Materials:	
Mill scale, 20 kg.....	.082
Lime, 30 kg.....	.087
Ferrosilicon, 6.5 kg.....	.485
Ferromanganese, 4 kg.....	.214
Power for cooling arrangements.....	.196
Repairs.....	.194
Total.....	8.822

To this must be added interest and the depreciation of the plant. Assuming 5 per cent interest and 10 per cent depreciation on \$13,380, the approximate cost of the plant, gives \$2,000 per year, which for 300 working days would give 41.2 cents per ton, so that the conversion of the fluid product from the basic converter into steel that can replace crucible steel for steel castings will cost \$9.23 per ton exclusive of the cost of the molten steel charged. This cost is figured on a basis of three men operating the furnace.

Cost of producing rail steel in a 3-phase Röchling-Rodenhauser furnace, from fluid steel.

	Cost per ton of product.
Raw materials, steel at \$15.54 per ton, and all fluxes.....	\$16.56
Power at 0.58 cent per kilowatt-hour:	
Heating up.....	.02
Refining.....	.72
Wages:	
For furnace.....	.16
For lining.....	.008
Lining materials.....	.072
Tools.....	.097
Repairs.....	.194
Amortization and interest.....	.168
Licenses.....	.582
Total.....	18.62

This estimate includes all items. If the cost of molten steel be disregarded, the refining cost, with power at 0.58 cent per kilowatt-hour, will be \$3.08 per ton.

* Loc. cit.

The cost of making casting steel of the composition given on page 105, from cold scrap in the Röchling-Rodenhauser furnace, at Lansdown, Pa., is estimated by the Crucible Steel Casting Co.,* as follows:

Cost of producing steel from cold scrap in the Röchling-Rodenhauser Furnace at Lansdowne, Pa.

	Cost per long ton of product.
Scrap.....	\$14.50
Oxidation loss, about 2 per cent.....	.29
	\$14.79
Power, 844 kilowatt-hours at 0.7 cent per kilowatt-hour.....	5.91
Fluxes.....	.76
Ore.....	.05
Labor.....	1.50
Tools, repairs, and bottom.....	.76
Air-cooling transformers.....	.12
Conversion cost.....	9.10
Interest, 5 per cent; depreciation, 10 per cent per ton.....	1.50
Cost of 1 gross ton (2,240 pounds) electric steel ready to pour.....	25.39

PROPERTIES OF ELECTRIC-FURNACE STEEL.

For many years all high-grade steels were manufactured by the crucible process, but since the advent of the electric furnace there has been a gradual adoption of that furnace for refining steel. For the complete refining of the highest grades of steel the use of the electric furnace is now thoroughly established in Europe. Any product that can be made by the crucible process can be made by the electric furnace, and in most cases with cheaper raw materials and at a lower cost. In the electric furnace complex alloy steels can be made with precision. The high temperatures attainable facilitate the reactions and alloys need not be used so largely for the purpose of removing gas. Very low carbon steels can be kept fluid at the high temperatures. Steels free from impurities and of great value for electrical apparatus can be made. With the electric furnace large castings can be made from one furnace, whereas in the crucible process steel from several crucibles must be used. For small castings, which require a very high grade metal free from slags and oxides, electrically refined steel is especially adapted. The electric furnace gives a metal of low or high carbon content as desired, hot enough to pour into thin molds and still free from slags and gases.

There is now a tendency among consumers of rail and structural steel to require a higher-grade steel at an increased price rather than steel of acid Bessemer or even of basic open-hearth grade at a lower price. With the high cost of power that now prevails throughout the steel centers of the United States the electric furnace can not

* Von Baur, C. H., "The Röchling-Rodenhauser furnace of the Crucible Steel Casting Co., Lansdowne, Pa. : Metall. Chem. Eng., vol. 11, 1913, p. 113.

compete profitably with either the acid Bessemer or the basic open-hearth process in manufacturing steel of like grade from pig iron. It is in combination with either of these processes that the electric furnace seems destined to be prominent in steel manufacture. The cost of superrefining in the electric furnace the molten steel from either of these processes, exclusive of the cost of the molten steel, varies from \$1.50 to \$2.25 per ton, depending on the cost of power and the impurities to be removed.

Experiments conducted by the United States Steel Corporation* during the past four years show that, as compared with the acid Bessemer and basic open-hearth processes, the electric process has the following advantages: A more complete removal of oxygen; the absence of oxides caused by the addition of silicon, manganese, etc.; the production of steel ingots of 8 tons weight and smaller that are practically free from segregation; reduction of the sulphur content to 0.005 per cent if desired; reduction of the phosphorus content to 0.005 per cent, as in the basic open-hearth process, but with complete removal of oxygen.

Acid Bessemer steel has been refined in the basic electric furnace, yielding steel of the following composition: Carbon, 0.55 per cent; manganese, 0.137 per cent; silicon, 0.13 per cent; sulphur, 0.017 per cent; and phosphorus, 0.022 per cent. Rails made from this steel are comparatively soft, but show less wear than Bessemer rails in the same track and subjected to the same service. About 5,600 tons of standard electric steel rails from electric-furnace steel have been in service in the United States for the past two years. These rails have been subjected to all sorts of weather and to temperatures as low as -52° F. It seems that rails made by the basic electric process can be made softer than by either the acid Bessemer or basic open-hearth processes and yet show highly satisfactory wearing qualities.

No steel rails made by the basic electric process in service in this country have yet broken. Electric-furnace steel of a given tensile strength has a slightly greater elongation than basic open-hearth steel and is somewhat denser than basic open-hearth or acid Bessemer steel.

Below are given the averages of some comparative tests of rails from steel made in a Röchling-Rodenhauser furnace and Bessemer rails. The tests were made at Volklingen, Germany, for the Prussian State Railway.

* Clark, E. B., Various types and applications of electric steel furnaces: Metall. Chem. Eng., vol. 10, 1912, p. 373.

Comparison of steel rails made in Röchling-Rodenhauser furnace with basic Bessemer steel rails.

Kind of steel.	Carbon.	Silicon.	Manganese.	Phosphorus.	Sulphur.	Tensile strength per square inch.	Elongation.	Contraction.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Pounds.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Electric.....	0.54	0.21	0.984	0.056	0.042	119,000	15.4	24.1
Bessemer.....	.36		1.127	.073		97,000	17.6	26.2

The results of some comparative tests, made at South Chicago, of electric-furnace steel for plates and basic open-hearth steel for plates were as follows:

Average ultimate strength and elongation of electric and open-hearth plate steels.^a

Electric.			Open-hearth.		
Carbon content.	Ultimate strength per square inch.	Elongation on 2 inches.	Carbon content.	Ultimate strength per square inch.	Elongation on 2 inches.
<i>Per cent.</i>	<i>Pounds.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Pounds.</i>	<i>Per cent.</i>
0.08	59,194	27.25	0.08	51,690	32.00
.12	64,080	26.05	.12	56,510	29.70
.16	69,220	25.25	.16	52,901	28.61
.20	72,853	22.82	.20	58,294	28.83
.24	69,540	23.12	.24	63,560	26.25

^a Osborne, G. C., The 15-ton Héroult furnace at the South Chicago works of the Illinois Steel Co.: *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 221.

The results show a 15.5 per cent increase in ultimate strength and 11.3 per cent decrease in elongation for electric steel, as compared with open-hearth plate steel of approximately the same chemical composition.

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SUGGESTIONS FOR A DUPLEX PROCESS FOR MAKING STEEL.

Inasmuch as greater capacity and efficiency can probably be obtained by working an open-hearth furnace and an electric furnace in conjunction, the following description and the costs of working a 5-ton tilting basic open-hearth furnace and a 2½-ton electric furnace in this manner, as obtained from data furnished by Electro-Metals, Ltd., of London, England, are submitted here.

Although a 5-ton open-hearth furnace is rather small for keeping a 2½-ton electric furnace working at full capacity, yet the following method of procedure would give the maximum efficiency with the given plant, and serves to illustrate the point.

The open-hearth furnace should be charged at the beginning of the week, say at 12 p. m. Sunday. The charge under the special circumstances will consist of 4 tons of scrap and 1½ tons of pig iron, with the usual fluxes. This being the first charge, it will probably be ready to tap into the electric furnace at 6 a. m. Previous to this time the electric furnace has been heated with coke and with current and is ready to take a charge of hot metal.

At 6 a. m. 2½ tons of partly refined metal are drawn from the open-hearth furnace and put in the electric furnace. The open-hearth furnace is then recharged by putting pig and scrap on the banks and slopes of the furnace and allowing the cold material to come to almost a fusing temperature. Then, when the new material is hot enough it should be pushed into the bath and worked in the usual manner. The metal should be fit to tap in three to four hours.

Meanwhile it has taken about one hour to heat the crude fluid metal and to dephosphorize it, two hours more being taken to desulphurize the charge, so that by 9 a. m. it would be ready to tap, and by 9.15 the electric furnace would be ready to take another charge.

It is quite possible that the open-hearth furnace may not be ready for another hour or more. If this be the case, about 1 ton of cold scrap may be charged into the electric furnace and melted, by which time the open-hearth furnace will be ready to tap. This time it will be necessary to tap only 1½ tons of metal into the electric furnace. Owing to the small recharge in the open-hearth furnace the charge will be completely melted and perhaps partly refined by the time the electric furnace has finished its work. The electric furnace then being ready again at, say, 12.30, a 2½-ton charge of very hot refined metal will be tapped into it immediately, with no waste of time.

This method of assisting the open-hearth furnace can be carried out when necessary; perhaps at every alternate charge. One will see that the process presents these advantages:

The ton of scrap melted in the electric furnace will not need any pig iron to accompany it, as would be the case in the open-hearth furnace. That means that one-fourth ton of pig iron at \$45 per ton is replaced by one-fourth ton of scrap at \$5 per ton. Here is a clear saving of \$10, which will pay for melting the ton of metal in the electric furnace many times over.

Although an "assisted" charge will take, say, $3\frac{1}{2}$ hours to work, the resulting gain of time for the open-hearth furnace will reduce the time of the succeeding electric-furnace charge to less than $2\frac{1}{2}$ hours and may even eliminate the dephosphorizing period in the electric furnace and thus reduce the time taken there to 2 hours.

The output by this method will be as follows:

One charge of $2\frac{1}{2}$ tons could be run every $3\frac{1}{2}$ hours, making 7 charges in 24 hours, or 42 casts per week of 6 days. This makes a weekly production of 105 tons, or a yearly production of 5,040 tons, allowing 48 weeks per year.

The cost of the process would be as follows:

Cost of producing steel by duplex process.

	Cost per ton.
Average power consumption, 420 kilowatt-hours, at 1.3 cents per kilowatt-hour.....	\$5. 46
Electrodes	. 56
Slags.....	. 62
Repairs to roof and lining.....	. 66
Labor, 3 men each shift.....	. 87
	8. 17

It should be noted that there is a saving of \$2 on every ton passing through the electric furnace by this method, owing to the use of scrap instead of pig iron.

DUPLEX PROCESS FOR LARGER FURNACE.

If the capacity of the open-hearth furnace be increased from 5 tons to, say, $7\frac{1}{2}$ tons, then it may be found that the following method of increasing the output can be employed.

The open-hearth furnace should be charged with, say, 8 tons of scrap and pig iron. The phosphorus is then reduced to the proper limit, after which a $2\frac{1}{2}$ -ton charge can be transferred to the electric furnace for completion of the process. The electric furnace will be ready again in 2 hours for another charge.

It should be quite possible to recharge the open-hearth furnace and have the bath dephosphorized again by the time the charge in the electric furnace is finished. In this way a charge should be ready every 2 to $2\frac{1}{2}$ hours.

The output will be as follows:

One charge of 2 tons could be run every 2 hours, making 12 charges in 24 hours, or 72 casts per week of 6 days. Therefore 130 to 140 tons would be produced in one week, or 6,200 to 6,700 tons in a year, allowing 48 working weeks to a year.

The cost of producing steel by this method would be as follows:

Cost of producing steel in duplex process, using 7½-ton furnace.

	Per ton of product.
Power consumption, 300 kilowatt-hours per ton, at 1.3 cent per kilowatt-hour-----	\$3. 90
Electrodes-----	. 50
Slags-----	. 37
Repairs to roof and lining-----	. 56
Labor, 3 men each shift-----	. 75
	6. 08

COST OF OPERATING AN ELECTRIC FURNACE ALONE.

For comparison with the foregoing schemes, the following figures for making steel in the electric furnace alone are given:

In a 2½-ton electric furnace 1 cast can be made in 6 hours, or 4 casts in a day, making an output of 24 casts, or 60 tons per week. On a basis of 48 working weeks in a year, the yearly output will be 48 times 60, or 2,880 tons.

The cost of producing steel in the simple electric process would be as follows:

Cost of producing steel in 2½-ton electric furnace.

	Cost per ton.
Power, 950 kilowatt-hours per ton, at 1.3 cents per kilowatt-hour-----	\$12. 35
Electrodes-----	. 87
Slags-----	. 62
Repairs, roof, lining, and dolomite-----	. 87
Labor, 3 men per shift-----	1. 50
	16. 21

Although the cost of operation is greater than that of the open-hearth, the cost per ton of the metal charged in the electric furnace is much less than that used in the open-hearth furnace, and this saving in cost should more than counterbalance the extra cost of working the electric furnace, not to mention the increased quality of the product.

A 5-ton electric furnace used for melting and refining would turn out 1 charge of 5 tons in 6 hours, or 4 casts in 24 hours. Twenty-four casts of 5 tons each, or 120 tons, would be produced in one week of six working days. The yearly production would be 5,760 tons, allowing 48 working weeks to a year.

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INDEX.

	Page.		Page.
A.		Charcoal, consumption of, in furnace	
Anderson furnace, description of	84	use of, as reducing agent	23, 24
Arc furnace, advantages of	120		47
characteristics of	74	Coke, use of, as reducing agent	27
power factor of	122	Colby, E. A., work of	59
Arc furnace and induction furnace,		Cowles, A. H., work of	59
comparison of	120	Cowles, E. H., work of	59
Arizona, deposits of iron ore in	39, 40	Crucible, for furnace, durability of	26, 27
Ashcroft, A. E., work of	51		
Austria-Hungary, production of electric-furnace steel in	72	D.	
B.		Dellwik, C., work of	33
Belgium, production of electric-furnace steel in	72	Deoxidation, process of, in refining steel	124
Bessemer steel, refining of	106, 107, 117, 118	Dommeldingen, Luxemburg, electric furnaces at, description of	119
Bessemer steel rails, tests of, results of	130	Domnarfvet, Sweden, furnaces used at, figures showing	14
Blast furnace, cost of operating	51, 52	Duplex process of producing steel, cost of	132, 133
reduction of ores in	29	description of	131, 132
Blast furnace and electric furnace, comparison of cost of	51	E.	
difference between	29	Electric furnace, cost of operating	51, 52
Bonn, Germany, electric furnaces at, description of	122	Electric furnace and blast furnace, comparison of cost of	51
plan and elevation of, figures showing	61, 62	difference between	29
Braintree, England, electric furnace at, description of	74, 94	Electric steel furnace, advantages of	129
products of	96	cost of operating	133
Bus-bar connections in Swedish furnace, figure showing	27	Electro-metals furnace. <i>See</i> Grönwall furnace.	
C.		Electrodes, arrangement of, in Swedish furnace, figure showing	27
Calcedined limestone. <i>See</i> Limestone.		consumption of	25
Calcium carbide, manufacture of, in electric furnace	59	reduction of	70
California, iron ore deposits in	39, 40	variations in	123
analyses of	41	cost of	123
practice of iron smelting in	50	early difficulties with, causes of	32
Canadian commission, investigations of	7-9, 63	graphite, use of	32, 45, 123
Carbon, for reduction of iron oxide, calculation of	30	steel, in Girod furnace, figure showing	79
proportion of, to CO ₂ , figure showing	34	Energy consumption. <i>See</i> Power consumption.	
solid, as reducing agent	49	England, production of electric-furnace steel in	72
Catani, R., work of	38	F.	
Chaplet furnace, description of	84	Ferranti, S. de, work of	58
		Ferro-alloys, manufacture of, in electric furnace	59

	Page.		Page.
France, production of electric-furnace steel in -----	72	Héroult furnace, power consumption of -----	65
Frick, Otto, work of -----	33, 35	Hirth furnace, description of -----	87
Frick furnace, description of -----	87		
Fuel, amount of, for blast furnace for electric furnace -----	29	I.	
Furnace gas, analysis of -----	22	Induction furnace, characteristics of power factor of -----	74 120, 122
circulation of -----	49	Induction furnace and arc furnace, comparison of -----	120
G.		Iron, pig, analysis of, from Héroult furnace -----	44
Gas, in electric furnace, circulation of, advantages of -----	49	cost of producing -----	57
method of improving -----	37, 38	quality of, from electric furnace -----	25
use of -----	36	variation in composition of -----	23
objections to -----	36	Iron-smelting plant, cost of, estimate of -----	54, 55, 56
Gas from electric furnace, value of -----	25	Iron ore, fine, smelting of -----	38
Germany, production of electric-furnace steel in -----	72	preheating, advantages of -----	38
Girod, P., work of -----	125	reduction of, conditions necessary for -----	33, 34
Girod furnace, chemical reactions in, figure showing -----	111	Italy, production of electric-furnace steel in -----	72
cost of -----	81		
cost of refining steel in -----	125, 126	J.	
current for, fluctuations of, figure showing -----	114, 115	Jones, C. C., work of -----	39, 40
description of -----	68, 77, 80, 81, 101		
elevation of, figure showing -----	68, 78, 80	K.	
output of, figure showing -----	115	Keller furnace, description of -----	66, 82
products of, properties of -----	116	figure showing -----	8, 67
products of, uses of -----	116	Keller furnace, experiments with -----	8, 9
voltage of, figure showing -----	115	principle of operation of -----	9
Girod steel plant, capacity of -----	96, 97	Kjellin, F. A., work of -----	62, 126, 127
description of -----	97, 98, 99	Kjellin furnace, description of -----	62, 63, 84, 85
products of -----	101	figure showing -----	63
Graphite electrodes. See Electrodes.		power consumption of -----	63, 64
Greene, A. E., work of -----	69	use of -----	84
Grönwall, A., experiments of -----	12	Knesche, J. A., work of -----	51
furnaces used by, figures showing -----	12, 13		
Grönwall furnace, cost of refining steel in -----	126	L.	
description of -----	82, 83	La Praz, France, electric furnace at, description of -----	89
figures showing -----	67, 68	figure showing -----	60
H.		operation of -----	89, 90
Haanel, Eugene, work of -----	7	steel produced by -----	90
Harbord, F. W., work of -----	8, 65	Lansdowne, Pa., electric furnace at, cost of producing steel in -----	128
Harden Paragon furnace, description of -----	84	description of -----	104, 105
Hering resistance furnace, description of -----	88	Lefler, J. A., work of -----	33, 37
figure showing -----	87	Limestone, calcined, use of, objections to -----	50
Herleinus A., work of -----	25	Lindblad, —, experiments of -----	12, 13
Héroult, Cal., electric furnace at, analysis of pig iron from -----	44	furnaces used by, figures showing -----	12, 13
description of -----	42, 43, 44, 46	Lining of furnace, rate of wear of -----	123
figures showing -----	42, 43		
Héroult, P. L. T., work of -----	8	M.	
Héroult furnace, cost of -----	75	Moissan, H., work of -----	59
figures showing -----	42, 43, 60		
description of -----	65, 74, 75, 76	N.	
electrode consumption of -----	65	Nathusius, Hans, work of -----	67, 68
		Nathusius furnace, description of -----	83, 84

	Page.		Page.
Nathusius furnace, figure showing--	69	Sheffield, England, electric furnace	
Nevada, deposits of iron ore in-----	39, 40	at, cost of refining steel	
Norway, production of electric-fur-		in-----	126
nace steel in-----	72	description of-----	91, 93, 103, 104
Nystrom, Erik, work of-----	10	figures showing-----	67, 68
		results of-----	92
O.		Siemens, W., work of-----	58
Oberhausen, Germany, electric fur-		Silver Lake Station, Cal., deposits of	
nace at, figure showing--	110	iron ore near-----	40
description of-----	109-111	Sintering, advantages of-----	38
products of, properties of--	116	Slags, in steel refining, composition	
uses of-----	116	of-----	112
reactions in, figure showing--	111	variation in-----	24
Odelberg, E., work of-----	25, 33, 53	function of-----	124
Open-hearth furnace, development of--	70	Soderburg furnace, description of--	84
process of smelting in-----	53	South Chicago, Ill., electric furnace	
		at, description of--	76, 105-107
P.		products of-----	107-108
Petroleum, crude, use of, as reduc-		Stalhane, ---, experiments of-----	12
ing agent-----	48	furnaces used by, figures show-	
Pig iron. <i>See</i> Iron.		ing-----	12, 13
"Pinch effect," definition of-----	88	Stassano, E., work of-----	7, 60-62
application of-----	88	Stassano furnace, description of--	81, 82
Power consumption, factors deter-		figure showing-----	61, 62
mining-----	120, 121	Steel, cost of production of-----	57, 133
influence of weight of charge on,		duplex process of producing,	
figure showing-----	114	cost of-----	132, 133
relation of to proportion of CO,		description of-----	131, 132
figure showing-----	35	electric furnace, characteristics	
variations in-----	23, 24, 120, 121	of-----	128, 129
Power factor, conditions governing--	121	yearly production of-----	72
definition of-----	121	refining of, reactions during--	111-113
method of regulating-----	122	Steel-refining plant, cost of, esti-	
variations in-----	122	mate of-----	54-56
Providence Mountains, Cal., iron ore		Stoble furnace, description of-----	84
in-----	40	Sulphur, removal of, in refining	
		steel-----	113, 124
R.		Sundberg, A. G., work of-----	11
Reducing agents. <i>See</i> Carbon, Char-		Sweden, iron ores in-----	11
coal, Coke, Petroleum.		production of electric-furnace	
Refining furnaces, method of-----	117	steel in-----	72
Richards, J. W., work of-----	29, 35, 36	Switzerland, production of electric-	
Robertson, T. D., work of-----	83	furnace steel in-----	72
Röchling-Rodenhauser furnace, cost		Switzerland, production of electric-	
of producing steel in--	127, 128	furnace steel in-----	72
description of-----	68, 69, 85, 86	Three-phase current, use of, discus-	
plan and elevation of, figure		sion of-----	69, 70
showing-----	69	Trollhättan, Sweden, furnace at, de-	
steel from, tests of-----	130	scription of-----	15-18
Russia, production of electric-fur-		figures showing-----	16, 17, 19
nace steel in-----	72	results of operation of-----	20,
Sault Ste. Marie, Ont., furnace at,			22, 23, 26
experiments at-----	10, 11	ore used in, analyses of-----	21
conclusions based on-----	11	furnace house at, figure showing--	15
figure showing-----	10	iron-smelting plant at, cost of	
Scotts Siding, Cal., deposits of iron		construction of-----	28
ore at-----	40		
Shaft, furnace, reduction of iron ore		U.	
in-----	33, 34, 35, 48, 49	Ugine, France, steel plant at, cost of	
volume of, to capacity of fur-		refining steel in-----	125, 126
nace-----	33	description of-----	97, 98, 99
		United States, production of electric-	
		furnace steel in-----	72

V.		W.	
	Page.		Page.
Van Norden, R. W., work of-----	45	Walker, W. R., work of-----	32
Völklingen, Germany, electric furnace at, cost of producing steel in-----	127	Wilson, T. L., work of-----	59
figure showing-----	69	Worcester, Mass., electric furnace at, description of-----	108
operation of-----	117		
products of-----	118	Y.	
relining of-----	117	Yngstrom, Lars., work of-----	12, 14, 29-31





Bulletin 70

Mineral Technology 2

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

JOSEPH A. HOLMES, DIRECTOR

A PRELIMINARY REPORT
ON
URANIUM, RADIUM, AND VANADIUM

BY

RICHARD B. MOORE AND KARL L. KITHIL



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CONTENTS.

	Page.
Preface.....	7
Introduction.....	9
Acknowledgments.....	9
The carnotite deposits of Colorado and Utah.....	9
Description of deposits.....	9
Coal Creek, Colo.....	10
Skull Creek, Colo.....	11
Split Mountain, Utah.....	13
Green River, Utah.....	13
Content of Green River ores.....	16
Table Mountain, Utah.....	16
Thompsons district, Utah.....	16
Paradox Valley and surrounding districts.....	18
Long Park, Colo.....	21
Club Ranch, Colo.....	24
Saucer Basin, Colo.....	25
East Paradox Valley, north side.....	26
East Paradox Valley, south side.....	26
Bull Canyon, Colo.....	28
McIntyre district.....	29
Hydraulic, Colo.....	29
Origin of the deposits.....	29
Mining methods and cost of mining.....	32
Transportation and prices.....	33
Concentration of ores.....	35
Necessity for concentration.....	35
Wet concentration.....	36
Results of wet concentration tests.....	37
Dry concentration.....	38
Cost of concentration.....	39
Possibility of chemical treatment.....	41
General statement.....	41
Pitchblende deposits.....	43
Description of deposits in the United States.....	43
North Carolina deposits.....	43
Colorado deposits.....	43
Kirk mine.....	43
German and Belcher mines.....	45
The Calhoun mine.....	46
The Wood mine.....	46
Waste of low-grade ore.....	46
Concentration of low-grade pitchblende ores.....	47
European pitchblende deposits.....	47
Uranium deposits in Portugal.....	49
Uranium ores in Australia.....	50
Vanadium from ores other than carnotite.....	51
San Miguel County, Colo., deposits.....	51
Huerfano County, Colo., deposits.....	52
Cutter, N. Mex., deposits.....	53
Eagle County, Colo., deposits.....	53
Deposits in other States.....	54
Deposits of patronite in Peru.....	55
Production of uranium, vanadium, and radium.....	56
Uses of vanadium, uranium, and radium.....	57
Uses of vanadium.....	57
Uses of uranium.....	58
Uses of radium.....	58

	Page.
Radium institutes.....	61
Market value of radium salts.....	62
Radioactive methods for testing ores.....	63
Method of using the electroscope for approximate determinations.....	64
Method of using the electroscope for exact determinations.....	66
Standardization of the electroscope.....	66
Correction for standardization.....	68
Method for exact determination of radium.....	68
Commercial methods of treatment of ores.....	69
Bleeker process for the recovery of vanadium.....	70
Haynes-Engle process for the recovery of uranium and vanadium.....	72
Koenig process for the recovery of vanadium.....	73
Fischer process of extracting vanadium from carnotite.....	73
Method used by Primos Chemical Co.....	74
Fleck method of extracting uranium and vanadium.....	74
Radcliff method for complex radium ores.....	75
Treatment of carnotite ores with nitric or hydrochloric acid.....	76
Bleeker process of separating uranium from vanadium.....	78
Process of extracting vanadic acid from copper vanadate.....	78
Austrian method of extracting radium from pitchblende.....	79
Analytical methods for uranium and vanadium.....	82
Methods for the analysis of carnotite.....	82
Method for the determination of uranium and vanadium.....	82
Method for the determination of vanadium.....	84
Method for the determination of uranium.....	84
Volumetric method for uranium in carnotite and vanadiferous uranium ores.....	85
Gravimetric method for vanadium and uranium in carnotite and other ores.....	88
Separation of alumina.....	89
Rapid method for the determination of vanadium in ores in the presence of iron.....	90
Minerals of uranium and vanadium.....	91
Publications on mineral technology and methods of mining.....	96
Index.....	99
Appendix.....	103

ILLUSTRATIONS.

	Page.
PLATE I. Part of uranium and vanadium ore region in Utah and Colorado....	10
II. A, Stratum of carnotite and vanadic sandstone of Thompsons deposits, Utah; B, Opening of North Star mine, Long Park, Colo., showing outcropping carnotite.....	16
III. A, Opening and dump of the Swindler mine, Long Park, Colo.; B, Cliff mine, Saucer Basin, Paradox Valley, showing 4-foot stratum of carnotite and vanadium ore between the points marked by the hat and the spade blade.....	24
IV. A, Cliff mine, Saucer Basin, Paradox Valley. Another view of 4-foot stratum of carnotite and vanadium ore; B, Electroscope for determining the radium content of an ore.....	26
FIGURE 1. Sketch map of Paradox Valley.....	22
2. Apparatus for separating emanation from uraninite.....	67

PREFACE TO SECOND EDITION.

Early in 1912, from information received by the Bureau of Mines, it became evident that large quantities of valuable material were being wasted in mining the rare-metal ores of the West. In pursuance of its endeavors to increase efficiency in the mining and treatment of mineral resources in the United States, the bureau assigned R. B. Moore, physical chemist, and K. L. Kithil, mineral technologist, to investigate the production of uranium and vanadium ores, the elimination of waste in mining, and the development of methods for converting the raw material into finished products. In the chemical side of the investigation R. B. Moore was assisted by C. F. Whittemore, to whom credit should be given for the analytical work appearing in this bulletin. Mr. Kithil has been engaged with Dr. Moore in a study of the general problem, and has given especial attention to the development of methods of mining and concentration.

These investigations have definitely shown that, although the Austrian Government has conserved its own resources of uranium and radium by purchasing the Joachimsthal mines and by carefully supervising pitchblende production, the deposits of radium-bearing minerals in the United States are being rapidly depleted by wasteful exploitation, chiefly for the benefit of foreign markets.

Seemingly the country has been quite unaware of the proportion of uranium ores sent abroad. According to the figures of the Bureau of Mines carnotite ores carrying 28.8 tons of uranium oxide were produced in 1912 and practically the entire amount was exported. The uranium content of the major part of the ore was equivalent to 2 to 3 per cent uranium oxide, as in that year no ore carrying less than 2 per cent could be marketed, owing to the cost of transportation. Radium equivalent to 11.43 grams of anhydrous radium bromide could have been obtained from the ore shipped in 1912.

In 1913, according to preliminary figures by the United States Geological Survey, 2,140 tons of ore was produced, of which 1,198 tons was shipped to works in this country and 942 tons was exported. However, owing to the cost of transportation, the richer ores were exported, so that actually more than one-half of the contained radium (equivalent to approximately 7.5 grams of anhydrous radium bromide) was sent abroad. One foreign company did not work its claims in the Paradox Valley during the greater part of the year because it was building a plant in Liverpool and it cost less to leave the ore in the mines than to take it out and store it in England. Only one American company has been preparing radium salts of high radioactivity and its product has only recently been offered for sale. According to a statement by its president, this company produced up to January 1, 1914, 2 grams of radium metal and expects to produce a much larger amount during the coming year.

It is improbable that all of the ores exported are now represented by finished product, but the actual production for 1912 and 1913 can not be much less than the quantity mentioned. The total quantity of uranium exported in 1911 was almost equal to that exported in 1912 and the ores are still being sold for treatment abroad. There can be no doubt that in 1912 and 1913 there was obtained from American ores more than twice and probably three times as much radium as from all other ores combined. In this connection it is interesting to know that a second American company is preparing radium salts, although it has not as yet marketed any of its finished product.

Since the first edition of this bulletin was written, prospecting in the Paradox Valley has been active. Many agents of foreign manufacturers have been in this country seeking supplies for shipment abroad, and the much stronger demand for the ores has caused prices to advance so that at present ore containing 2 per cent of uranium oxide readily brings \$80 per ton at the railroad at Placerville, Colo. Correspondingly higher prices are obtained abroad, and some private producers who were able to contract for considerable quantities report even higher prices.

Owing to the interest aroused and to the proposed withdrawal of radium-bearing lands, there has been a decided increase in prospecting and some important new finds have been made. Although many claims have been staked in different parts of the radium-bearing region, the most important finds have been at the summit of Big Canyon at the head of Lisbon Valley, Utah, just across the State line from the McIntyre district, Colo. Other important finds are reported in the Henry Mountains about 100 miles southeast of Green River and in a district about 7 miles northeast of Monticello, Utah. Development work has been carried on at Gateway and also in the western part of the Paradox Valley, which was the original scene of the main operations.

In mining uranium ore, for every ton marketed there is some 5 tons of low-grade material thrown on the dump and much more is left in the mine, its commercial exploitation awaiting only a feasible process of concentration. Mr. Kithil has shown that elutriation can be applied to both uranium and vanadium ores and that a large proportion of the valuable material now going to waste can be obtained in marketable condition. If water is not available, air separators may be used. A concentrating mill is now being planned and will soon be constructed in connection with the work of the National Radium Institute and preliminary laboratory experiments lead to the belief that important results will be obtained whereby much of the low-grade ore not previously available may be concentrated.

In 1913 lower grade ores were available for shipment abroad than in 1912 and there appears at present to be a demand for material even as low as $1\frac{1}{2}$ per cent uranium oxide.

The uranium deposits of Colorado and Utah are being depleted rapidly by foreign exploitation and it would seem almost a patriotic duty to develop an industry to retain the radium in America. So large a demand for radium has sprung up, owing to publication, by unquestioned authority, of statements as to its curative power in certain forms of cancer, that there are seemingly no supplies for immediate delivery.

Quoted present (January, 1914) prices for radium bromide, chloride, or sulphate for future delivery are as follows: Purity, 1 to 60 per cent, \$120 per mg. metallic radium; 60 to 75 per cent, \$140 per mg. metallic radium; 75 to 90 per cent, \$160 per mg. metallic radium. At \$120 per mg. for radium metal in salts of 60 per cent purity, the value of the radium in American ores in 1912 amounts to a little more than \$790,000 and in 1913 to about \$1,050,000, so that mining and separation of the ore can be considered only as one of our smaller industries even if the ore be treated at home. But the fact should be noted that a large part of this radium went into foreign hands and opened up to foreign medicine and science opportunities for investigation and treatment of malignant disease that have been denied to our own people except by repurchasing manufactured radium compounds at almost prohibitive prices.

France, Austria, England, and Germany have their radium institutes fostered by their Governments or by philanthropic foundations. During the last year extensive appropriations have been made abroad for the purchase of radium and mesothorium, showing the great demand that has arisen for these materials in the treatment of malignant disease. Some of these purchases have been made by Government appropriation and some by public subscription. The following list may be of interest.

Appropriations by German cities for purchase of radium.

City.	Sum raised.	To be purchased.		Purchase price.
		Quantity.	Mineral.	
	Marks.	Milligrams.		Marks per milligram.
Lepzig.....	288,800	600	76 per cent radium chloride.....	472
Berlin.....	242,000	600	57 per cent radium bromide.....	370
Do.....		200	Mesothorium.....	a 10
Chemnitz.....		150	Radium bromide.....	388
Dresden.....		200	do.....	388
Do.....		225	Radium chloride.....	458.5
Essen.....		125	60 per cent radium chloride.....	427
Dortmund.....	39,000	100	Radium bromide.....	
Munich.....	60,000	200	do.....	
Hanover.....	28,000	100	do.....	
Frankfort.....	200,000	600	do.....	
Hamburg.....	200,000			
Bremen.....	140,000	400	Radium bromide and mesothorium.....	
Lubeck.....		300	do.....	

a Monthly lease price.

The list is incomplete as Prussia has appropriated 370,000 marks for the purchase of 1 gram of radium, and many German cities of less than 80,000 inhabitants have also made appropriations and are attempting to procure radium for the treatment of cancerous growths.

In this country the National Radium Institute has been founded by Dr. Howard A. Kelly, of Baltimore and Dr. James Douglas of New York City, and a cooperative agreement with the National Radium Institute and the Bureau of Mines has been announced. Under this agreement the bureau obtains the opportunity of a scientific and technologic study of the mining and concentration of carnotite ores and of the most efficient methods of obtaining radium, vanadium, and uranium therefrom, with a view to increased efficiency and the prevention of waste.

A plant is already under construction at Denver, Colo., and the lease of 16 claims containing carnotite obtained from the Crucible Steel Mining & Milling Co. Over 100 tons of high-grade carnotite is already ground and ready for treatment as soon as the plant is completed. Concentration experiments will be conducted and if successful will be applied to reducing wastes now taking place. Within a year mill operations should make results certain and the extraction of ore and production of radium will be continued on a larger scale. All processes, details of apparatus and plant, and general information gained will be published for the benefit of the people. The institute has been formed for the special purpose of procuring enough radium to conduct extensive experiments in radium therapy with special reference to the curing of cancer; also it expects to conduct experiments in connection with the physical characteristics and chemical effects of radium rays. Hospital facilities in Boston and New York are already supplied.

The activities of the institute are sure to be of benefit to the prospector and miner by providing a greater demand for his already rare ore; to the plant operator by developing methods and by creating a larger market for his product; and to the people by assisting and possibly by succeeding in controlling the most malignant of diseases. The radium produced is intended for the institute's own use and will consequently remain at home.

C. L. PARSONS,
Chief, Division of Mineral Technology.

A PRELIMINARY REPORT ON URANIUM, RADIUM, AND VANADIUM.

By RICHARD B. MOORE and KARL L. KITHIL.

INTRODUCTION.

This bulletin presents a summary of available information regarding the sources of uranium, radium, and vanadium, the methods used in treating the ores, and the uses of the finished products. In particular the paper describes the ores found in the United States, giving especial attention to those characteristics of the ores and the conditions of their occurrence that affect mining and treatment.

ACKNOWLEDGMENTS.

In the preparation of this bulletin a number of persons rendered assistance by furnishing information and by making it possible for us to visit the carnotite deposits in western Colorado and eastern Utah at a time of the year when without this assistance it would have been difficult to obtain satisfactory results. We desire to express our thanks to the following persons: Henry Hall, W. L. Cummings, T. V. Curran, K. D. Hequembourg, David Taylor, H. C. Brown, Forbes Rickard, George W. Alsdorf, Dr. Herman Fleck, Orr J. Adams, C. S. Cherrington, Newt. Stewart, O. B. Willmarth, Angus Cameron, George A. Head, and Lorimer Bros.

In addition, we also wish to express our cordial thanks to Charles F. Whittemore for a number of chemical analyses made by him and for other valuable assistance.

THE CARNOTITE DEPOSITS OF COLORADO AND UTAH.

DESCRIPTION OF DEPOSITS.

Up to the present the carnotite deposits of Colorado and Utah have excited little interest outside of those two States, although they are the most important uranium-bearing deposits known in the world. During the fall of 1912 the authors were able to visit the more important localities, which are scattered over a considerable area, as shown

by the accompanying map (Pl. I). A description of the deposits visited follows:

COAL CREEK, COLO.

The Coal Creek deposits of carnotite are 14 miles northeast of Meeker, the county seat of Rio Blanco County, Colo. The claims, 10 in number, are reached by wagon road and lie east of Spurlock Ranch, just beyond Henry Ranch. Little more than prospect work has been done and it is difficult to estimate the extent of the deposits.

The carnotite occurs in the lower bed of a group of beds of massive white sandstone that underlies the Dakota sandstone (conglomerate). The outcrops are on or near the hogback ridge that is formed by the lowest and most massive bed. For further details of the geology, the reader is referred to Gale's work.^a

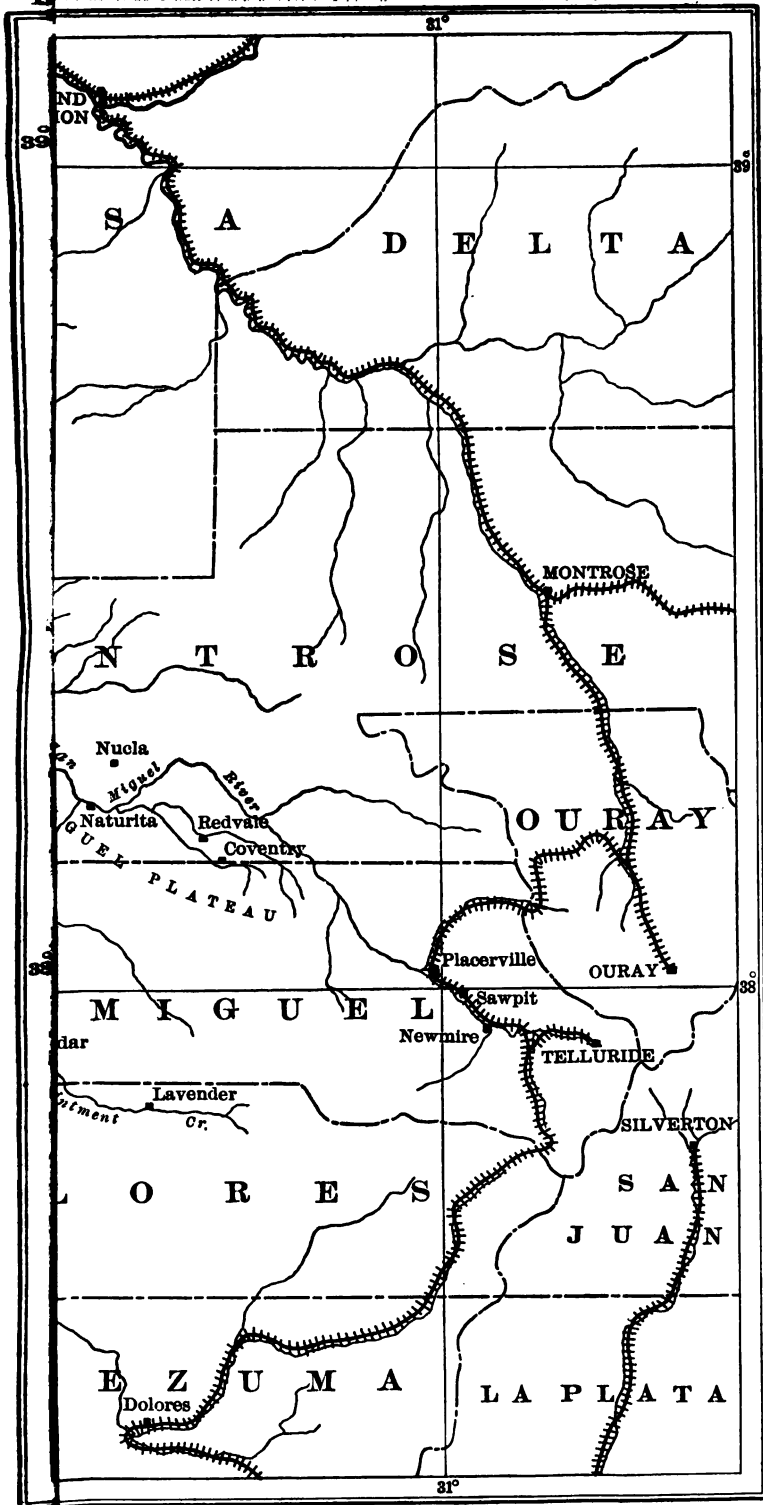
On the Caywood claims nine prospect pits were examined, of which several show no signs of carnotite. No. 1 shows none; in No. 2, about 1,000 feet above Coal Creek and 300 feet higher than No. 1, is an exposure of sandstone colored a light green in streaks and layers for a thickness of about 6 feet from the top; the entire rock is somewhat impregnated with the color, which is due to chromium. In this pit no carnotite ore is exposed. Some petrified wood was found at its bottom, also showing chromium stains.

The next pit, No. 3, just below No. 2, and southwest of it on the same hill, contains a petrified tree about 12 inches in diameter lying halfway across the pit. Cracks and interstices in this tree, which is rusty brown, are filled with powdery yellow carnotite. The sandstone for 1 foot below the tree carries good ore. Under the grass roots the sandstone is soft and rather heavily impregnated with carnotite. Farther below, in the white sandstone, a brownish-yellow streak occurs, underlying which is a somewhat richer yellow material a few inches in thickness. On the right side of the pit, near the bottom, is a brownish-colored rock containing vanadium.

About 15 feet northwest from pit No. 3 is another opening in which is exposed a part of the petrified tree that is exposed in pit No. 3. Interstices in the tree, as in pit No. 3, are filled with rich yellow carnotite. In this prospect hole no other signs of carnotite were observed. The next pit, No. 5, about 12 feet from the one just described, shows only some yellow-colored sandstone at its mouth. In the sandstone below is a layer of black to dark-brown, much decomposed, material, rich in vanadium.

East and about 1,000 feet higher up on the ridge, at an elevation of about 9,500 feet, and near the top of the highest peak of the mountain, are the so-called Elkhorn claims. There are two prospect pits, in the first of which, at a depth of 9 feet measured from the

^a Gale, H. S., Carnotite in Rio Blanco County, Colo.: U. S. Geol. Survey Bull. 315, 1907, pp. 110-117.



1. The first part of the document is a list of names and titles, including "The Hon. Mr. Justice" and "The Hon. Mr. Justice".

2. The second part of the document is a list of names and titles, including "The Hon. Mr. Justice" and "The Hon. Mr. Justice".

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top, a streak of dark-brown to black material about 12 inches thick is embedded in the sandstone that underlies the conglomerate (Dakota sandstone). No showing of carnotite was seen in the pit. The second opening, about 10 feet north of the first, shows the same occurrence of the dark-brown material near the top; about 10 feet below the surface is a layer of sandstone strongly impregnated with carnotite; this layer thickens to about 13 inches. The ore here is more like the carnotite-bearing sandstone of Paradox Valley in the southern part of the State. Scattered over the surface of the hillside are pieces of petrified wood.

On the west side of the same mountain, but below pits Nos. 4 and 5, is the Caywood No. 6 claim. Here the yellow color is noticeable only where the sandstone has been exposed to the air. Beneath the surface no stains are visible. The pieces of ore lying on the dump and exposed to the air show the discoloration to a marked degree. A bed of carbonaceous material 6 to 14 inches thick lies in the sandstone about 3 feet below the surface.

Several hundred feet to the southwest are the Caywood Nos. 7 and 8 claims. Pit No. 7 has no indication of ore. In No. 8 the sandstone is slightly discolored with yellow carnotite. Southeastward and down the same hill is a tunnel (No. 9) containing only thin streaks of yellow ore. At the bottom of the face of the tunnel is a streak of black carbonaceous material 2 inches thick.

The development at this place is not sufficient to give any definite idea of the extent of the deposits. They are patchy and it is far from certain that the ore will be found in commercial quantities.

The freight rate for ore is about 75 cents per 100 pounds from Meeker to Rifle, and about 25 to 50 cents per 100 pounds from the deposit to Meeker.

SKULL CREEK, COLO.

Another occurrence of carnotite and other vanadium ores is found in the northwestern part of Colorado. The deposits are in Skull Creek Basin in Routt County, 65 miles west of Meeker, 42 miles east of Jensen, Utah, and about 18 miles east of the Colorado-Utah line. The deposits are best reached from Mack, Colo., a station on the Denver & Rio Grande Railroad, by the Uintah Railroad (narrow gage) to Watson, and thence by stage to Vernal, Utah, where an outfit can be obtained to bring the traveler by way of Jensen to Skull Creek.

These deposits are between Wolf and Red Wash Creeks and are found in the foothills of the Blue Mountains, in the hogbacks of white sandstone.^a Red clays and shales underlie the carnotite and

^a Gale, H. S., Carnotite and associated minerals in western Routt County, Colo.: U. S. Geol. Survey Bull. 340, 1908, p. 269.

vanadium bearing strata. Between the white sandstone which carries the carnotite and the conglomerate (Dakota sandstone) there is a series of beds composed of variegated clays and marls with limestone layers.

Several of the claims were examined. The Lookout claim is on a hill, locally termed Uranium Hill, on the west side of Skull Creek Basin. The north side of the hill shows layers of green-stained white sandstone. The green material also occurs as incrustations and in many places is cemented with gypsum. The outcrops also show larger quantities of dark-brown and black materials, many of which carry vanadium. In two of the drifts the surface shows a beautiful color effect from green, dark-brown, black, and light-brown layers. The green color is due to malachite with many specks of blue azurite. These stains can be traced along the north side of the mountain toward the west for a considerable distance. Yellow coloring is rare.

Other claims are on top of Skull Creek Mountain, on the east side of Skull Creek Basin, about 500 feet above the valley. These deposits are difficult to reach as the steep ascent of an eroded crevice in the face of the rock on the south side of the mountain must be used. From the first pit in the Little Emma claim on top of the hogback 10 tons of carnotite was shipped to Germany, but the ore was found to be too low in U_3O_8 . From all appearances it seems that the prospector has opened up the drift just above the carnotite-bearing horizon that lies under the excavations. The rock pitches about 40° SW. The sandstone that carries the carnotite also contains copper. Overlying the carnotite ore is much green-stained sandstone, its color being largely due to malachite. Underlying the carnotite is a thin streak of black, soft, coal-like material, and also an ore with green, as well as yellow, stains due to carnotite and probably malachite. The following layers were observed on the hanging wall of the opened drift, from top to bottom:

White material (calcite).

Sandstone with incrustation of bright greenish and yellow tint (1 to 2 inches).

Dark-brown vanadium and uranium ore with yellow specks (2 to 5 inches).

Thin sandy layer, dark brown to black (vanadium):

White sandstone with green impregnation (several feet).

Earthy blackish-brown streak carrying vanadium.

Several other claims lie along the hogback ridge for several miles in a westerly direction in the same group of strata. These were not examined owing to lack of time. The ores seem to be mostly vanadium and copper ores containing some uranium, the valuable minerals being in the white sandstone underlying the conglomerate frequently referred to as Dakota sandstone.

From the development work done it is impossible to obtain any correct idea as to the extent of these deposits.

SPLIT MOUNTAIN, UTAH.

A deposit of uranium ore had been reported at Little Split Mountain in the vicinity of Island Park, near the Horse Shoe Bend of Green River, Uinta County, Utah, but on investigation no such occurrence was found.

The road from Jensen by way of Rainbow Park to Island Park is rough, especially between Rainbow and Island Parks, across the mesa. The locality visited is about 30 miles northeast of Jensen and the trip takes an entire day. From Island Park a trail 6 miles long leads over steep mountains and through ravines to the claims on the north side of Green River. There are also some claims on the south side of the river, which cuts the mountain in two.

No uranium or vanadium ore was found. Only the claims on the north side of the river were investigated, there being several prospect pits, drifts, and shafts. Near the base of the mountain, just above the river, is a large mass of limestone breccia; overlying this is a decomposed shaly rock carrying iron but no carnotite. The layers are 20 feet thick and are thinly foliated. Just above the shale is a soft, brownish-black carbonaceous streak, 6 inches thick. Overlying this streak is decomposed material of shaly appearance, somewhat similar to that mentioned above. This is overlain by 3 feet of white sandstone that shows narrow veins of malachite and azurite and is well impregnated with these carbonates. Above the sandstone is 6 to 8 inches of black coal-like material. At the bottom of the drift is a black shale carrying streaks of white sandstone about 1 inch thick that show small round green and blue specks of the copper carbonates.

The south side of the river was not examined, but in the openings on the claims the same stratification is said to be visible. The ore-bearing bed continues on that side of the river, but 500 feet up the mountain side, showing that the river has cut it in two. There is said to be a 40-foot shaft and an open cut 75 feet long.

GREEN RIVER, UTAH.

West and south of Green River, Utah, the San Raphael swell reaches a height of several hundred feet above the surrounding plain. The main axis runs almost north and south. The eastern side is much eroded and the beds of sandstone, shale, and conglomerate are tilted at various angles. San Raphael River cuts eastward across the northern part of the swell, turns south through the valley that divides the "swells" from the "reefs," and finally again takes an eastward course through the "reefs" to join Green River.

The "reefs" dip at an average angle of about 30°. They are cut at right angles to the direction of the valley which divides them from the "swells" by a series of nearly parallel gulleys. Many of the gulleys are only 100 or 200 yards apart, although they cut the edges of the "reefs" for several miles. The conditions are clearly favorable for prospecting, the beds being freely exposed on both sides of the valley of San Raphael River and also in the gulleys. Little prospecting and practically no development has been attempted except in connection with ore bodies thus exposed.

Some of the vanadium and uranium deposits were visited by Boutwell ^a and later by Hess.^b

Most of the carnotite deposits are 10 to 12 miles southwest of the town of Green River. There is a good wagon road with no steep grades, so that in hauling ore a round trip can be made in one day. On the railroad 6 miles from the mines is a siding, but several deep gulleys have to be crossed to reach it and the road is not good, hence the ore is preferably taken to Green River.

The majority of the deposits are exposed in the gulleys that traverse the "reefs." The carnotite is always in a rather coarse sandstone overlain with fine conglomerate. Much petrified wood is exposed, also bones and other fossils. As at Meeker, the carnotite stains are conspicuous about the wood, much of which is heavily impregnated. The yellow ore is found mostly in or near the wood or in cracks in the sandstone, although both the sandstone and the darker ores are lightly impregnated in many places. Yellow ore such as occurs in Paradox Valley is largely absent here, most of the ores being dark colored. They may be divided into four general types—the yellow carnotite, found mainly in cracks; a dark-brown siliceous ore impregnated with carnotite; a black ore much of which is associated with carbonaceous material and some of which carries stains of carnotite; and a grayish-brown laminated sandy shale, rich in vanadium and carrying some uranium. Parts of the deposits show mixed ores, but the majority of them can be classified as above.

Many samples that show little or no carnotite turn yellow on exposure to the air for several weeks. This yellowing takes place quickly if the ore is heated, as by laying it on the top of a stove. Such a test frequently shows uranium in ores not suspected of containing it, and as the test is well adapted to fieldwork, it merits a wider use.

The principal claims are the Lorimer and the Forsman. At the former is a good, permanent camp, and 8 to 12 men have been constantly employed during the year. Mining on the other claims has not been carried on systematically, ore being mined in small quanti-

^a Boutwell, J. M., Vanadium and uranium in southeastern Utah: U. S. Geol. Survey Bull. 260, 1905, p. 201.

^b Hess, F. L., Carnotite near Green River, Utah: U. S. Geol. Survey Bull. 530, 1912, p. 18.

ties from time to time and kept in Green River until enough for a car has accumulated.

About a mile south of the Lorimer claims is the Little Ruth claim. Below 4 feet of fine conglomerate there is 3 to 5 feet of sandstone, then about 5 inches of dark vanadium ore that carries some carnotite. This can be traced through occasional outcrops for a distance of 25 yards. At places the sandstone is impregnated with carnotite. Below the pay ore is conglomerate.

North of the Little Ruth is the Napoleon claim, on which is a cut 8 to 10 feet long. A 2-foot layer of sandstone shows the color of ore, and there are some layers of black carbonaceous material. The ore as a whole is low grade.

South of the Little Ruth is the Vernon Junior claim. A cut shows conglomerate at the top, underlain by fine sandstone, 2 inches of black, rich-looking vanadium ore, and 15 inches of shaly sandstone, showing yellow stains of carnotite but evidently a low-grade ore. The sandstone above the black ore is lightly impregnated with carnotite.

Still farther south is the Wardvern claim, where ore has been taken out of a cut about 20 yards long. A 12-inch band of sandstone is impregnated with carnotite along the whole length of the cut. Above this sandstone band is a thin streak of black ore.

The Monayunk is one of the Lorimer claims. The ore body is exposed for about 40 feet, but it pinches out at the ends. At the thickest part color shows over a width of 3 feet. The sandstone has stains of yellow both above and below a streak of black ore which has some wood in it and appears to be partly carbonized.

On the Grand View claim is a cave about 50 feet long that extends across the ore-bearing zone. Black vanadium ore shows in patches and stains at places along the roof.

Some of the best ore at Green River is on the Melrose Discovery claim. The ore is exposed for a distance of 120 feet. Below the conglomerate is sandstone which overlies an ore body averaging 2 feet in width. Below the ore is conglomerate, sandstone, and a second ore body 8 inches thick.

Six miles south of the Lorimer claims are the Loveless-Forsman claims. These are situated in a less broken country at the foot of the "reefs," the ore outcropping at several places from under loose and soft sand. This sand carries carnotite at grass roots and is a low-grade ore.

About 3 miles north of these claims and on the west side of the river several tons of ore have been mined at the Morris claims, the only ones that have been worked on that side of the river. Most of the prospecting has seemingly been confined to the "reefs," little time having been given to looking for ore along the "swells,"

CONTENT OF GREEN RIVER ORES.

Shippers from Green River have had trouble in connection with analyses of their ores. Analyses of six lots of ore, a total of 36,100 pounds, showed an average content of 2.53 per cent V_2O_5 and 3.33 per cent U_3O_8 . These results, however, were questioned, and the ore was delayed in transit for some time pending a settlement. There is no doubt that little, if any, ore shipped out of Green River has assayed over 2 per cent U_3O_8 . Most of the ore carries between 1 and 2 per cent. The vanadium content in the above analyses is probably correct. Some shipments have assayed as high as 8 per cent V_2O_5 . Owing to the ore being low in uranium, it must be picked carefully in order to get ore of shipping grade, 2 per cent. Picking causes considerable waste; moreover, efficient picking is difficult, owing to the character of the ore, many of the dark samples that seemingly show little yellow carnotite being richer in uranium than some highly colored samples.

The operators at Green River have the advantage of a lower combined haulage and freight rate than those at any other locality producing carnotite. The transportation charges from the Lorimer camp to New York are \$17.50 per ton in carload lots. The only other locality having an approximately similar rate is Thompsons, Utah. It costs more than \$17.50 to haul ore from any point in Paradox Valley to the railway at Placerville, Colo.

During the year 1912 346 tons of ore was shipped from Green River, Thompsons, and Cisco, the only points in Utah that shipped ore. A part of this carried less than 1 per cent U_3O_8 and the returns did not equal expenses. It is doubtful whether any of the ore carried 2 per cent U_3O_8 ; most of it carried about 1.5 per cent. Approximately 125 tons additional was mined and stored but not shipped.

TABLE MOUNTAIN, UTAH.

Forty-five miles south of Green River is Table Mountain, where are deposits of red calcium vanadate, and carnotite in beds of sandstone several feet thick carrying both uranium and vanadium. Lumps of asphaltic material containing uranium and vanadium are frequently found.

THOMPSONS DISTRICT, UTAH.

The Vanadium Ores Mining & Milling Co. claims are about 16 miles southeast of Thompsons, a station on the Denver & Rio Grande Railroad, in Grand County, Utah.

The deposits (Pl. II, A) are best reached from Thompsons over a fairly level road that after having traversed the surrounding foothills crosses a long stretch of desert lands in a southeasterly direction.



A. STRATUM OF CARNOTITE AND VANADIC SANDSTONE (SEE CROSSES) OF THOMPSONS DEPOSITS, 16 MILES SOUTHEAST OF THOMPSONS, UTAH.



B. OPENING OF NORTH STAR MINE, LONG PARK, COLO., SHOWING OUTCROPPING CARNOTITE.

About 13 miles from Thompsons the desert changes into a rugged hilly country. The road leads through small winding canyons to the camp, which is erected in an opening of a canyon near a spring.

The deposits are $1\frac{1}{2}$ miles north and west of the camp and cover a considerable area.

During the night of our arrival in camp heavy snow fell, making a thorough examination of the claims impossible except where the surface of diggings and outcrops was exposed.

There was an exposed outcrop on Telluride No. 8 claim along the perpendicular face of a hillside. The yellow impregnations in the sandstone can be followed along the side of the hill for a considerable distance. A quantity of low-grade ore is lying on the dump. The carnotite-bearing zone of the sandstone is at this place 8 to 9 inches thick. Overlying this is a strip of carbonaceous material. Two feet below the carnotite there is a layer of what looks to be roscoelite 2 feet thick.

At different places in the sandstone are small pockets of red vanadium ore, probably calcium vanadate. This ore shows imprints of fossil-fern leaves and stems.

Some richer carnotite was found on claim No. 5, where at various points is a grayish rock, the cracks and fissures of which are filled with incrustations of rich yellow mineral. Some ore at this place is also found in pockets. In the same deposit there is a black shaly material the surface of which, after exposure to the air, turns decidedly yellow.

A decomposed sandstone is found in streaks, small layers, and pockets; it is greenish-yellow and carries both vanadium and uranium in small quantities.

Claim No. 1 shows at several places a thin streak of sandstone impregnated with carnotite. There are small pockets of a mixture of ores in which the carnotite, as well as the dark vanadium ore, is embedded in and between thin layers of crystallized gypsum. The little of this ore in sight is high grade.

Near the spring on claim No. 1 and near the top of the hill there is a small quantity of coal-like material in a hole about 3 by 10 inches. This material is highly radioactive, and after having all of the adhering carnotite removed by washing and crushing it is quite as radioactive as before. It may therefore carry uranium oxide.

On top of the hill at various points in the same canyon there are several outcrops of uranium and vanadium ores. These ores are embedded in a greenish-colored sandstone which constitutes a considerable part of the rock on the hillside. At one locality, at a distance of 1 to 3 feet above the lower stratum of the sandstone, the

yellow carnotite is found in a loose sand. It is of low grade, but might make a milling ore.

The deposits form a connecting link between those of the San Raphael swell on the west and those at Richardson on the southeast and, with several breaks, ultimately reach the districts near Hydraulic, Club Ranch, Saucer Basin, and Long Park.

The Thompsons deposits are almost flat-bedded, but there is an unconformity and some faulting. At the time of the visit no work was being done, but during 1912 some ore was shipped, which, however, was held up during transit, as it contained less than 2 per cent of uranium oxide. The vanadium content was relatively high. This trouble can be obviated by more careful sorting. The ores in this district as a whole are mostly of low grade, but can probably be concentrated. It is difficult to estimate the possible amount available. Mining has been confined to the best outcrops, and the low-grade ore from hand sorting is thrown on the dump.

The water supply at or near the mines in this district is limited, and timber has to be brought from a distance. Grand River, near its confluence with the Dolores, is 8 miles west of the camp.

Other deposits of uranium ore in Utah are at Richardson, Fruita, and Moab. From Moab some low-grade material has been shipped.

PARADOX VALLEY AND SURROUNDING DISTRICTS.

The carnotite deposits in San Miguel and Montrose Counties, Colo., have been known for a number of years. As far back as 1881 Andrew J. Talbert mined some ore and sent it to Leadville, where it was tested for gold, silver, and copper. The report stated that it carried \$5 gold per ton. In 1897 Gordon Kimball and Thomas Lothin sent specimens to the Smithsonian Institution at Washington, D. C., and were informed that the mineral contained uranium. Shortly afterwards Kimball and Lothine mined 10 tons of ore and shipped it to Denver where it was sold for \$2,700. In 1899 Poulot and Voilleque, two Frenchmen, visited Paradox Valley, collected specimens, and sent them to Friedel and Cumenge in France, who announced in the French journals the existence of a new mineral which they named carnotite and described as potassium urano-vanadate.

Poulot and Voilleque in 1900 began operating at a copper mine at Cashin in Paradox Valley, where they used leaching vats to extract the uranium. Shortly afterwards they built a small mill in the McIntyre district, south of Paradox Valley. In this project they had the cooperation of James McBride of Burton, Mich. The mill ran until 1902, and during this time produced about 15,000 pounds of uranium oxide. The mill was started again in 1903 by the Western Refining Co., but ran only until 1904. Shortly afterwards the Dolores Refining Co. built a new mill a short distance from the old one, but

after running for some years it, too, shut down. The concentrate, which was obtained by the Engle process, retained uranium and vanadium only, not the radium. In addition to this concentrate, some ore was shipped during this period. In 1912 the American Rare Metals Co. acquired the mill of the Dolores Refining Co. and is now operating.

Although the McIntyre district was the scene of some activity, little was done in the Paradox district until the formation of the General Vanadium Co. in 1909. That company began work early in 1910, in the same year that the Standard Chemical Co., of Pittsburgh, Pa., entered the field, and these two companies are now the largest operators in the district. The former have at present about 60 claims and the latter 90. The total number of claims filed is between 500 and 600, but a number of these are practically worthless. With reviving interest in the deposits claims were rapidly located, and during the last two years the output of ore has been the largest in the history of the industry.

Hillebrand and Ransome^a have written concerning these deposits. Fleck and Haldane^b have given a very complete account of the deposits in the Paradox and surrounding districts during the early stages of the work. Hess^c has written a short account of some of the claims. In addition, a number of articles have been published by the mining journals. However, the field of operations has, to a large extent, changed since the articles mentioned were written, old deposits having been worked out and new ones opened.

The Paradox Valley is at the western end of the high plateau that slopes westward from Norwood to the eastern base of the La Sal Mountains. The Dolores River enters on the south and runs across the valley to the northeast, instead of following the valley lengthwise along the natural grade. Hence the name "Paradox."

The deposits are confined to a well-defined area. The eastern boundary can be represented by a line drawn from a point a little east of the junction of Dolores River and Disappointment Creek on the south, through a point 6 miles west of Naturita, and thence due north to San Miguel River. The western boundary is the La Sal Mountains which extend north beyond Uranium almost to Gateway. The total length of this area from north to south is about 40 miles and the width 20 miles. On the western side of the La Sal Range, in Utah, are the Moab deposits, and north of them are those of Richardson and Thompsons. Farther west are Green River, Table Mountain, Pahreah, and the other Utah uranium fields. If it were not for the

^a Hillebrand, W. F., and Ransome, F. L., Carnotite and associated minerals in western Colorado: *Am. Jour. Sci.*, vol. 10, p. 134.

^b Fleck, Herman, and Haldane, W. G., A study of the uranium and vanadium belts of southern Colorado: *Rept. State Bureau of Mines, Colo.*, 1905-6, pp. 47-115.

^c Hess, F. L., Carnotite near Green River, Utah: *U. S. Geol. Survey Bull.* 530, 1911, pp. 161-164.

break caused by the La Sal Mountains, a close connection might be drawn between the Utah and Colorado deposits. The main difference is that the former are of lower grade and more widely scattered.

The principal localities where work has been done during the last two years are Long Park, Club Ranch, Saucer Basin, Hydraulic, north and south sides of East Paradox, Bull Canyon, and the McIntyre district. Roc Creek, the west end of Paradox Valley, and other localities have also furnished some ore.

The ores of the Paradox district differ in many respects from those of Utah, chiefly in carrying larger proportions of carnotite, and, as a rule, are more yellow. Not only do they carry more uranium, but also on the average more vanadium, although individual shipments from Utah might seem higher in vanadium than the average from the Paradox district. The following analyses of samples from several shipments of ore from Long Park during the past year will give some idea of the kind of ore that is being handled:

Analyses of ores from the Paradox district.

Constituent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
U ₃ O ₈	3.14	2.83	3.35	2.75	2.48	2.91	3.43	3.54	2.95
V ₂ O ₅	5.33	5.14	5.81	5.08	5.09	4.25	13.63	6.66	5.93

These analyses show a vanadium and uranium content higher than the average, a considerable part of the ore shipped containing as low as 2 per cent U₃O₈, whereas a large amount of lower-grade material is left in the mine or thrown on the dump. Occasionally a few hundred pounds of high-grade ore carrying 15 or 20 per cent U₃O₈ is obtained from "bug holes."^a All the shipments from Paradox Valley and the surrounding districts during the past year would probably average about 2½ per cent U₃O₈ and between 3 and 4 per cent V₂O₅.

The most typical ore is a sandstone so impregnated with yellow carnotite that the color is decidedly noticeable and containing small kidneys of brown sandy clay. The kidneys constitute a considerable part of some of the ore; in many cases they are thinly scattered through the sandstone. It seems to be generally accepted among the operators that the kidneys are rich in vanadium. The samples we have tested show vanadium. Although ore of the character mentioned is widely distributed in the Paradox and adjacent districts and constitutes a large part of the ore shipped, it is by no means the only ore of commercial importance. Indeed the variety of the types of ore here and also in Utah is one of the interesting features of the uranium and vanadium deposits. We have had time to test only a few of the

^a A local term meaning a small pocket, lined with rich ore, in the formation. It is probably derived from "vug."

large number of samples taken. There are dark-blue, brown, and black vanadium ores, the dark-blue ores being lustrous when first mined and usually carrying uranium; high-grade carnotite in "bug holes," so soft that it can be molded in the fingers; the same kind of ore crystallized with gypsum; and red calcium vanadate, some in radiated form, and some mixed with carnotite and blue vanadium ore. Much of the very low-grade ore on exposure to the air weathers to a green, rose, or yellow color, or to all three colors intermingled. In many places several ores of different types are mixed in an intricate mass; in other places the sandstone is impregnated along the lines of stratification and there are alternate layers of carnotite and dark vanadium ore. It can readily be seen that it requires considerable skill and experience to sort such ores properly, especially as the sorting is done on the basis of the uranium content and not the vanadium content.

The deposits are invariably pockets, many of which, however, are of considerable size; 50 tons of shipping ore from a single claim is not unusual. Several claims have yielded more than this. Many of the pockets are exposed in the sides of the canyons, but at other places, notably Long Park, development work has to be done. The ore is found in a light-colored sandstone overlain in places with shale and conglomerates. According to Hillebrand and Ransome this is the McElmo formation.^a The conglomerate is seen especially well in crossing from Club Ranch to the Saucer Basin. Below the McElmo formation lies a fine-grained sandstone (La Plata), and below this the Dolores or Red Beds.

LONG PARK, COLO.

The distance from Placerville, Colo., the nearest railroad point, to Long Park is 56 miles (fig. 1), which can be divided as follows: Forty miles to Naturita, the terminus of the stage route, 4 miles to Coke Ovens, where the Standard Chemical Co. has its headquarters and a large ore house, and thence 12 miles to Long Park. Coke Ovens is at the head of Paradox Valley, and the road from that point gradually climbs the escarpment on the north side of the valley to a height of about 1,000 feet, then drops 200 feet on the other side into Long Park, which is 6,500 feet above sea level. The park is about 3 miles long and one-half mile wide. The surface is gently undulating, with a gradual rise on the south side to about 200 feet, the north side being more precipitous.

The southern side of the western end of the park is traversed by two shallow, rather narrow valleys running from west to east. In the first of these are the Turner claims. At the time of our visit a new pocket had just been exposed. The ore was found at a depth of

^a Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado: U. S. Geol. Survey Bull. 262, 1905, p. 11.

2 feet in a shallow cut in the side of the spur between the two valleys. The carnotite-bearing streak in the sandstone was about 8 inches wide where first cut; it thinned to 2 inches and then widened again to about 8 inches. There is considerable gypsum present.

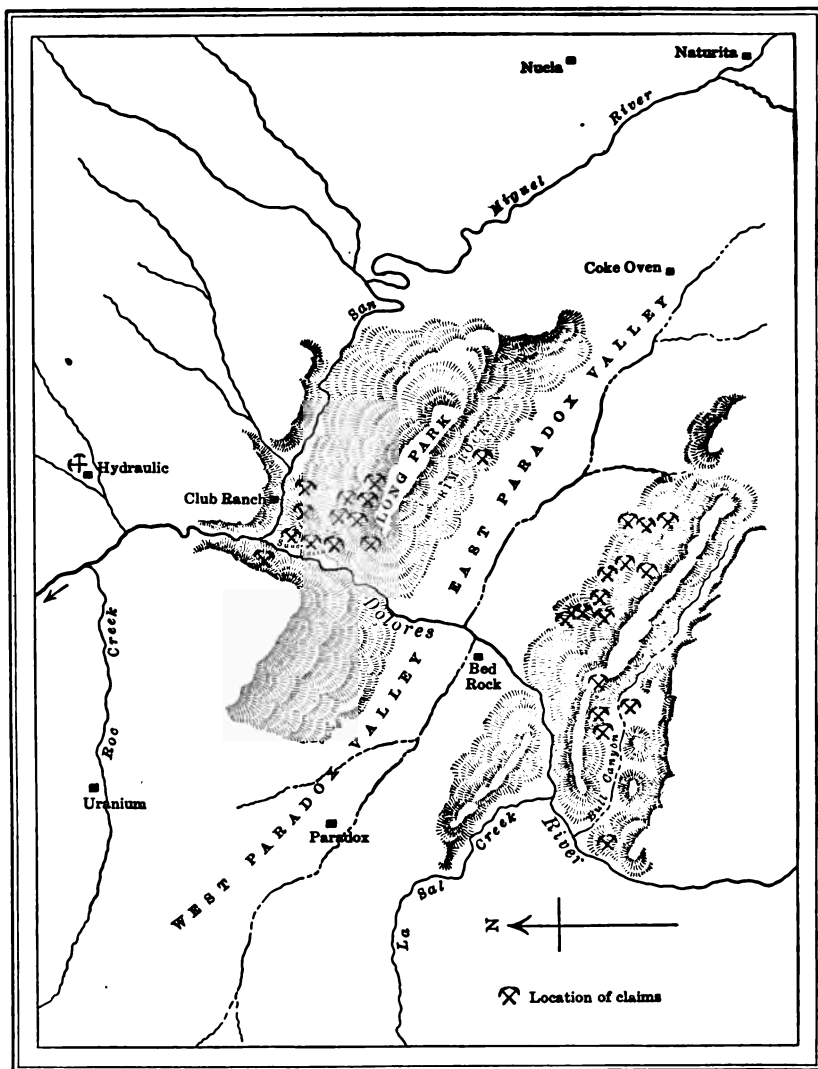


FIGURE 1.—Sketch map of Paradox Valley.

On the south side of the same valley and opposite the newly found pocket is a large cut that has furnished much shipping ore. The dip of the beds is about 20° S. The ore remaining is both yellow and black in color, the average being a grade that would require careful hand sorting. There is a large quantity of milling ore on the dump.

Farther south is the Turner No. 4 claim, which shows 5 or 6 feet of massive sandstone underlain with 3 or 4 feet of hard shale partly impregnated with carnotite and containing small pockets of richer ore. Some dark vanadium ore is also present. The underlying sandstone is gray.

Several other openings on this property have yielded shipping ore, but were not being worked at the time of our visit.

In the second valley are the Curran claims, 13 in number. They are mostly situated along the north side of the lower half of the valley. Several promising openings are in the face of the low cliff overhanging the valley (Pl. II, *B*). Both tunnels and drifts are being driven and also open cuts. On the Cripple Creek claim the ore appears to have collected in a V-shaped pocket, the country rock being white sandstone. At the top of the pocket there are 15 inches of a dark-gray vanadium ore, and below this 4 feet of good uranium ore consisting of carnotite mingled with a bluish-black vanadium mineral, which lies in the carnotite in small local masses and stringers. The pocket dips in the direction of an ore-bearing zone observed in a short adit a little to the east and at a lower level, indicating that one may be a continuation of the other.

The Swindler claim (Pl. III, *A*) is to the west of the Cripple Creek. Here nearly 200 feet of tunnel work has been done, the ore having been followed from the outcrop on the face of the cliff. The ore is very similar to that of the Cripple Creek, the width varying at different points. On the dump is considerable low-grade material which represents the waste after the ore has been hand picked. Vanadium ore low in uranium is left in the mine. A number of sacks of rich ore has been obtained from "bug holes," or rich pockets.

At the upper end of the same valley is the camp of the Radium Extraction Co. which, with the General Vanadium Co., is a subsidiary company of the International Vanadium Co. of Liverpool, England. The latter is closely associated with the George Blackwell Sons Co. of Liverpool.

The claims of the Radium Extraction Co. are situated on the hog-back north of the camp. There are three drifts; two of them are driven northwest and seem to be in the same body of ore. Lower on the hillside another drift has been driven to cut the ore body which seemed to strike in that direction. Over the ore is a thin bed of sandstone colored red by numerous small spots of iron oxide. Below it is 8 inches of medium-grade ore, then 1 to 2 inches of high-grade ore, and below this 2 feet more of medium ore. The ore dips 10° N. and strikes 10° E. Underlying it is a dark sandstone probably carrying a little vanadium, but this is not being mined. The ore was discovered 12 feet in the drift, from which point the ore body broadened to the dimensions stated.

In drifting for ore the miners follow the upper surface of the gray sandstone, and the appearance of splotches of dark-red and almost black ore is taken as an indication that carnotite is near.

In the lower part of the carnotite-bearing strata are black patches of what appears to be good vanadium ore. This is thrown on the dump.

The ore mined is hand sorted into three grades, high, middle, and low, in an average proportion of 3 sacks of high to 15 of middle and 23 of low grade.

The beds show an unconformity and some are faulted. Mining and development are carried on at the same time.

The Crucible Steel Co. claims, 7 in number, are also in Long Park. Most of them are near the Radium Extraction Co. claims, on the same hill. All assessment work has been completed, but no mining for shipping purposes has been carried on. Most of the prospect holes show good values. In ore in a ravine about 200 yards southeast of the Radium Extraction Co. claims are several cylindrical sleeve-like pockets, or "bug holes." They are 2 to 4 inches in diameter and are filled with canary-yellow high-grade carnotite. The ore when first removed is so soft that it can be molded in the hand, but on exposure to the air it becomes hard and brittle. At these pockets the ore is surrounded by a hard envelope of ferruginous sandstone about 2 inches thick, outside of which the sandstone is soft for some distance. Such a hard coating is not found at all "bug holes," the ore at some being surrounded by soft sandstone. Occasionally the envelope is largely gypsum, showing that calcium sulphate was carried by the original mineral-bearing solution and crystallized out first.

In several places the ore-bearing sandstone contains carbonaceous material that has a yellow coating. This carbonaceous material is radioactive, but contains no vanadium.

There is another good showing of ore at an outcrop in a ledge about 75 feet above the spring close to the camp of the Radium Extraction Co.

The Primos Chemical Co. claims, 7 in number, are in and around Long Park. Most of them are along the edge of a cliff, 3 miles north of the camp of Curran and the Radium Extraction Co. Assessment work only is being done. Little ore is exposed and this is rather low-grade vanadium ore.

In addition in and near Long Park are the Standard Chemical Co., Patton, Furr, Stone, and Rossler claims.

CLUB RANCH, COLO.

The Club Ranch deposits are about 7 miles from Long Park. There being no road between the two places, it is necessary to follow a rather difficult trail across rough country. The ore must be packed



A. OPENING AND DUMP (BELOW ARROW) OF THE SWINDLER MINE, LONG PARK, COLO.



B. CLIFF MINE, SAUCER BASIN, PARADOX VALLEY, SHOWING 4-FOOT STRATUM OF CARNOTITE AND VANADIUM ORE BETWEEN THE POINTS MARKED BY THE HAT AND THE SPADE BLADE.

from the Club Ranch claims on burros to Long Park, from which point it is sent in wagons to Placerville. The main operator at Club Ranch is the Standard Chemical Co., which has an ore house at Long Park for convenient transfer of ore from burro to wagon. The deposits can also be reached from Nucla. A road passes near Club Ranch, where, after fording the river, one can take a steep trail up the cliff on the south side of San Miguel River.

There are a number of open pits, from some of which considerable ore has been taken. Several tons of low-grade material lie on the dump, much of which probably carries 1 per cent U_3O_8 . The ore seems to strike south and dip east. The overburden is usually 3 to 4 feet thick. The overlying sandstone, as well as the sandstone that carries the ore, contains thin seams of crystalline gypsum. Considerable work has been done here and it was stated that over 900 sacks of ore had been mined and packed from the claims. Nothing was being done, however, at the time of our visit.

The formation seems to be uniform throughout. The ore is in more regular layers than in the Long Park deposits, and there is more blue, black, and gray vanadium ore and less carnotite, the latter lying between layers of vanadium ore. In many respects the deposits are similar to those at Thompsons and San Raphael swell.

The overlying sandstone is filled with small, round, reddish-brown specks of iron oxide, similar to that in Long Park.

SAUCER BASIN, COLO.

Three miles south of the camp of the Standard Chemical Co. at Club Ranch are the four Wilson claims, which constitute the Cliff mine. The mine is in the rim rock and lies 500 feet above the floor of Saucer Basin and 6,100 feet above sea level. To the west, at a distance of $1\frac{1}{2}$ miles, is the Dolores River, which cuts across the canyon at its western end after having traversed Paradox Valley.

A layer of carnotite-bearing sandstone extends 500 feet along the face of the cliff (Pls. III, *B*, and IV, *A*). It has more the appearance of a vein than any other similar deposit in the Paradox district. The pocket of ore is almost horizontal at the surface, but at 15 feet in, it begins to dip slightly to the north. At the top is white sandstone, then 4 to 6 inches of carnotite ore showing thin dark-gray and black layers of vanadium ore. Below this is 3 feet of similar vanadium ore that carries carnotite and in places narrow bands of decomposed quartzite. There seem to be two ore bodies with 3 feet of barren sandstone between; perhaps there is only one body, which is faulted. In places the barren sandstone carried a little carnotite, the result of leaching.

Most of the material exposed is low grade, but it has been found necessary to work one claim only thus far, owing to the fact that the drift has been in pay ore nearly all the time. This claim has been

one of the heaviest producers in the history of the industry. Some prospect work is now being done on the property.

On a ledge a short distance from the mine a forge had been in use for some time. The heat from the fire caused some of the surface of the near-by sandstone, previously uncolored, to show a distinct stain of carnotite.

At various times considerable high-grade ore has been obtained from "bug holes." A quantity of low-grade milling ore lies on the dump, but seems to be mixed with waste rock. A rather large quantity of low-grade ore has been left in the mine. At one place the deposit widened to 14 feet.

The overlying sandstone makes a good roof and little timbering has to be done. Pillars of ore are left in the workings, or, where rich ore is encountered, pillars are built of waste rock.

By building an aerial railway from the mine openings over a low hill to the west the ore could be carried to the banks of Dolores River, where a small mill could be erected for concentrating it, or the ore might be treated nearer the mine by piping water from the river. At present the ore is packed on burros 10 miles, by way of Club Ranch camp, to Long Park, thence hauled in wagons to Placerville.

EAST PARADOX VALLEY, NORTH SIDE.

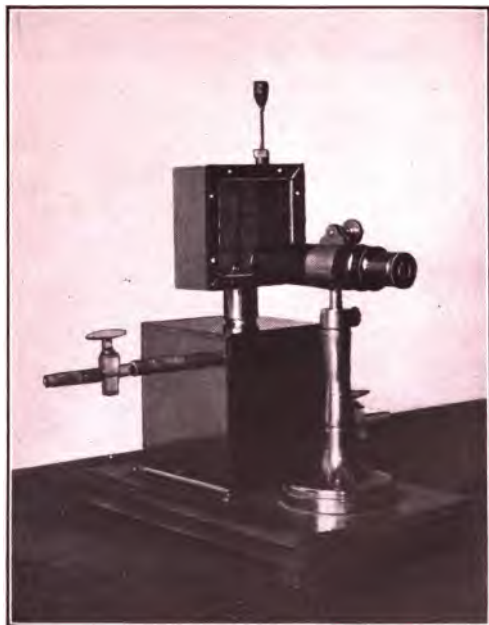
Two hundred feet above the road leading from Coke Ovens to Long Park and about halfway between these two places are the Jacobs and McKeever claims, on which are two prospect cuts and two or three short tunnels. The variety of ores in a small space is larger here than at any other locality in the Paradox district—carnotite; calcium vanadate; carbonaceous material that is highly radioactive and is also rich in vanadium; blue, black, and gray ores of vanadium; and "bug holes" containing carnotite. The ores are intermingled rather than deposited in layers. Although the ore body is 2 to 3 feet thick, it is difficult to estimate the extent of the deposits, as not enough development work has been done. A recent shipment of 13 tons from this place carried 3.43 per cent U_3O_8 and 13.66 per cent V_2O_5 .

EAST PARADOX VALLEY, SOUTH SIDE.

Along the road into Paradox Valley from Coke Ovens, the first mine on the ridge that bounds the valley on the south is the Thunderbolt, from which considerable ore has been taken. The mine is several hundred feet above the bottom of the valley and can be reached only by trail from the main road. The ore must be packed on burros to the foot of the hill and there transferred to wagons. About 2 miles down the valley, on a continuation of the same ridge at a height of 700 feet, is the main camp of the General Vanadium



A. CLIFF MINE, SAUCER BASIN, PARADOX VALLEY. ANOTHER VIEW OF 4-FOOT STRATUM OF CARNOTITE AND VANADIUM ORE.



B. ELECTROSCOPE FOR DETERMINING THE RADIUM CONTENT OF AN ORE.



Co. Operations had ceased for the winter on December 1, 1912. From most of the deposits at this place all the ore that can be shipped at a profit under present conditions has been taken. There is, however, a large quantity of low-grade ore in place besides what is on the dumps. No water is available for concentrating, the supply for the camp having to be brought by team and burro at a cost of \$1.25 per barrel.

The Jo Dandy, behind the camp and a little higher on the ridge, was until recently a heavy producer, but is now nearly exhausted. There are six or seven openings to the workings. The ore is in many ways similar to that of the McKeever prospect on the opposite side of the valley. A "bug hole" contained some blue-black and red vanadium ore. The surface of the sandstone on the sides of one of the openings is stained dark green, light green, purple, and orange yellow, the latter tint predominating, but the sandstone is almost white below. Gypsum is found throughout the mine, layers from an eighth to a quarter inch thick being common, and surrounds some of the "bug holes." There is considerable low-grade ore left in the mine, besides what is on the dump.

West of the Jo Dandy is the Blackbourne claim, where a number of drifts follow an ore body that outcrops. All shipping ore has been removed from these drifts, but bands of low-grade ore, about 2 feet thick, outcrop almost continuously for a distance of more than 200 yards.

The Valley View, Opera Box, and Kent Smith claims lie west of the Blackbourne in the order named. The Standard Chemical Co. owns one, the General Vanadium Co. owns the other two. The three claims are 200 feet above the Blackbourne, the duplication of the ore-bearing zone being due to a "slide," which is visible all the way to the Monogram claim, 2 miles farther west. In the Valley View claim the ore body dips 10°, then 30°, and then pinches out.

The ore left in these claims is low-grade vanadium ore that carries a little carnotite, usually as thin layers. All three claims have been good producers of vanadium and have also yielded a considerable quantity of carnotite.

Two miles west are the Monogram, Greenback and other Standard Chemical Co. claims, which are reached from the camp by a trail along the ridge. Also a steep trail, along which ore is packed, ascends from the valley below. The Monogram and Greenback No. 3 are both on a second "slide" and about 150 feet below the mines on the first "slide." Much of the overburden has been removed and practically all of the ore taken out. West of the Monogram and at the same level is the Quarter Circle, where a cut into the hillside yielded 100 sacks of good ore. Greenback No. 2, above Greenback No. 3, was nonproductive and Greenback No. 1, above Greenback

No. 2, yielded only 80 sacks of ore which was not high grade. West of the Greenback claims and above the Quarter Circle is the Bob-tail, on which two open cuts produced some medium-grade ore. The ore body, however, pinched out. Farther west is the Last Chance, and west of this claim are the Happy Thought, Little Tom, and Annie May claims. Some good ore is exposed on the Annie May, although 150 sacks have been taken out. The overburden is heavier than on most of the claims. On practically all there seems to be a direct relation between the thickness of the overburden and the quantity and grade of the ore, the latter improving where the overburden is thicker. This relation is discussed in the chapter on the origin of the deposits. The ore body in the Annie May is in the form of a series of sharply-dipping steps. On the Holliday, below the Annie May, is a tunnel that connects with a 30-foot shaft from the Annie May. There was no ore in the tunnel, but some carnotite showed in a crosscut. Some medium-grade ore was taken from the Wilson, to the southwest. Still farther southwest are the Jasper and Greystone claims on which assessment work only has been done.

On all of these claims the ore is in pockets. Much of the shipping ore has already been removed.

The Standard Chemical Co. has operated many of its claims during the year, but practically all of the ore mined in 1912 was stored at its headquarters at Coke Ovens.

BULL CANYON, COLO.

Bull Canyon lies nearly due south of Paradox Valley and 8 miles from the Monogram mine. A rather rough trail ascends the ridge on the south side of Paradox Valley for 500 feet and then descends more than 1,000 feet into a long, narrow canyon which ultimately leads into Bull Canyon. The scenery is wild and the topography rugged. The canyons are bordered by high steep cliffs of red sandstone capped with the white sandstone in which are the uranium deposits. Some ore is taken out by way of Monogram camp on pack animals and transferred to wagons in Paradox Valley, and some is packed several miles over another trail and a road to Redvale, whence the ore is taken to Placerville.

The camp at the Cummings claims is well situated, and about 12 men have been employed. Operations began in October, 1912. The claims are rather scattered. The Black Fox is about 4 miles down the canyon from the camp. The outcrops are in the rim rock 150 feet above the canyon. There is a relatively thick overburden of sandstone containing four thin streaks of blue clay. On the south side of the ore body a sandstone roll cuts across at an angle of 45°. The ore extends to this roll, but not beyond it. At the top of the pocket

a "bug hole" 3 inches in diameter carried vanadium ore, dark blue, with some red splotches. The main body of ore, 3 feet thick, was largely a mixture of blue and greenish-yellow vanadium and uranium ores carrying red streaks of calcium vanadate. The dip is about 10° N. About 8 feet north of the pocket the calcium vanadate disappears and the ore is yellowish green on top and black mingled with blue below. At this point the ore body was 1 foot thick, but the bottom had not been exposed. Forty sacks had already been taken from the prospect.

Two and one-half miles northeast of the Cummings camp, on the west side of the canyon and some 300 feet above the bottom of the valley, is the Boot Leg claim, on which some carnotite is exposed. Seventy feet higher is the Widow claim, on which prospect work has just begun. There are indications of carnotite in a greenish-yellow sandstone that outcrops on the hillside. Overlying this sandstone is another which at first sight looks much like decomposed granite. At the Fawn claim, across the canyon from the Widow, some carnotite is exposed, but only prospect work has been done.

On the Bob-o-link and Sundown claims, about 1 mile from the Cummings camp, is a fair showing of both carnotite and a dark vanadium ore. Some ore has been shipped from the Bob-o-link.

The Wedding Bell claim, near the Black Fox, has produced some good ore by careful sorting.

In Bull Canyon are also the General Vanadium Co., Saunders, and Cloud claims.

MCINTYRE DISTRICT.

The McIntyre district, which lies south of Bull Canyon and was the scene of some of the earliest mining for carnotite, was not visited by the writers. In this region the American Rare Metals Co., of Denver, has its plant. We are informed that the work done has made only small inroads on the ore supply. The larger part of the ore is rather low grade. Shipments have to be made by way of Dolores.

HYDRAULIC, COLO.

The General Vanadium Co. during the year opened a number of prospects at Hydraulic, and is now working at that place an aerial tramway across the river, saving a haul of several miles.

ORIGIN OF THE DEPOSITS.

It is difficult to form a definite opinion as to the origin of the carnotite and vanadium deposits. Hillebrand and Ransome^a show that the ores must have been carried to their present position and that the vanadium and uranium compounds could not have been the

^a Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado. U. S. Geol. Survey Bull. 262, 1905, p. 17.

original cementing material of the quartz grains, but in all probability locally replaced the calcite that forms the matrix of the ordinary light-colored sandstones in which the ores occur. They express the opinion that the carnotite resulted from local concentration of material already in the sandstone, and that its deposition as carnotite was under conditions determined by proximity to the surface and probably was partly dependent on a semiarid climate.

Hillebrand ^a has shown that in small amounts vanadium is widely distributed in sandstones, limestones, and igneous rocks. Occasionally the proportion of vanadium in a sandstone is such that the material may be worked commercially at a profit. In the roscoelite-bearing sandstone deposits around Newmire, Colo., the ore mined contains on an average $1\frac{1}{2}$ per cent V_2O_5 , and in places as much as $2\frac{1}{2}$ per cent V_2O_5 . In the newly discovered silver and vanadium bearing sandstone in Eagle County, Colo., some of the ore contains $2\frac{1}{2}$ per cent V_2O_5 . It is therefore not difficult to state a possible origin of the vanadium that has been concentrated in the carnotite deposits. Almost invariably vanadium minerals are associated with the carnotite.

To explain the origin of the uranium is more difficult. No pitchblende, even in small quantities, has been found near these deposits, Gilpin County, several hundred miles away, being the nearest locality where pitchblende occurs. In the absence of definite knowledge as to any deposit from which uranium could have been derived, it seems reasonable to believe that the uranium came from sandstone overlying or underlying the ore bodies, having been leached from these sandstones and concentrated with the vanadium. In some cases it has been noticed that the ore is underlain by an impervious blue clay, which may have been a factor in determining the concentration of the uranium compounds. This is particularly true at Thompsons.

The so-called "bug holes" appear to have escaped the notice of Hillebrand and Ransome. Many of these holes are 30 to 40 feet long and 2 to 5 inches in diameter; the walls are usually incrustated with quartz or gypsum. Almost invariably these holes run downward at a slight angle into the upper parts of an ore body, although a few enter the lower part of a deposit, and end abruptly in the ore. They are filled with high-grade ore, usually carnotite, although in some the blue and black ores of vanadium predominate. The other end of these "bug holes" opens into a funnel-shaped mass of soft sandstone, heavily impregnated with ore, that grades into the country rock. The appearance of one of these "veins" is that of a funnel with a long stem. Undoubtedly these holes represent channels through which ore-bearing solutions were transported. How far

^a Hillebrand, W. F., Distribution and quantitative occurrence of vanadium and molybdenum in rocks of the United States: *Am. Jour. Sci.*, ser. 4, vol. 6, 1898, pp. 209-216.

the ore-bearing solution traveled and whence it came are questions more difficult to answer.

We collected a number of samples of the overlying sandstones, which appeared to be perfectly free from uranium stains of any kind. Determinations of the radioactivity of some of these samples are given below:

Radioactivity of samples of sandstone.

1. Sandstone 3 feet above ore, Wilson mine, Saucer Basin: 1 gram contains 3.7×10^{-12} gram of radium.
2. Sandstone 2 feet above ore, Long Park: 1 gram contains 261.5×10^{-12} gram of radium.
3. Sandstone 3 feet above ore, Skull Creek: 1 gram contains 262.8×10^{-12} gram of radium.
4. Sandstone 3 feet above ore, Black Fox claim, Bull Canyon: 1 gram contains 9.4×10^{-12} gram of radium.
5. Sandstone from sandstone roll, Black Fox claim, Bull Canyon: 1 gram contains 105.4×10^{-12} gram of radium.
6. Sandstone 3 feet above ore, Telluride No. 8, Thompsons: 1 gram contains 2.9×10^{-12} gram of radium.
7. Sandstone 1 foot above ore, Telluride No. 8, Thompsons: 1 gram contains 23.5×10^{-12} gram of radium.

There are very few sandstones in other localities having a radioactivity of more than 2×10^{-12} gram of radium. That of the sandstone of the Desert of Sahara is 0.4×10^{-12} gram of radium. A fair average of all sandstones is probably about 1×10^{-12} gram of radium. The results given above show that the radioactivity of the sandstones tested is 3 to 262 times the average. On account of this large variation it is difficult to draw definite conclusions, but the figures show that in many cases the sandstone 2 to 3 feet above the ore contains a quantity of radium, and therefore of uranium, many times the normal. Sample 5 was taken from a sandstone roll that cut off rich ore on the Black Fox claim. This sandstone was favorably situated to receive leachings from the rich ore, and yet it is less than half as active as some samples of sandstone collected 2 and 3 feet above ore. On the other hand, the fact that on the south side of East Paradox Valley the thickness and richness of the ore deposits seem to vary directly with the thickness of the overburden appears to lend some weight to the theory of downward enrichment. Further examination of the country rock will probably throw light on the origin of the ore deposits. The results already obtained, however, seem to indicate that the uranium was disseminated in the sandstone country rock and has been concentrated in ore bodies by the action of water, the "bug holes," in some cases at least, acting as channels for the ore-bearing solutions.

MINING METHODS AND COST OF MINING.

Simple methods have been employed in mining carnotite, and so far most of the surface deposits have been worked by open cuts. At a few places drifts and adits have been run to follow ore into a hillside. Prospecting with drills has not been done, so far as we could ascertain, but drills will have to be used later when the surface deposits have been exhausted. At the Cliff mine in Saucer Basin a number of drifts and crosscuts have been driven into the cliff side; in most of them there is some timbering, but as a rule the roof is firm enough to stand without support. At another claim a deep shaft is sunk to a drift, both shaft and drift being well timbered.

Only hand drills are used in mining. The usual explosive is a "40 per cent" dynamite, which does not shatter the rock too much. Excessive shattering of the rock would cause the loss of much valuable mineral as fine dust. To reduce losses in the dust and fine ore, care must be taken in sorting.

The exact cost of mining is not easy to ascertain, as detailed accounts are not kept by the average miner, but a safe estimate, based on information received from various persons, is that the cost including hand sorting, exceeds \$20 and perhaps averages \$30 per ton of shipping ore. In one case the mining, sorting, and sacking of 1 ton of carefully mined shipping ore required the services of 5 men for 3 days, the men being paid \$3 each per day. Therefore, the labor cost of this ton of shipping ore was \$45. To this cost must be added the cost of sacks, which is \$3 per ton of ore when single bags are used. The mining and sorting cost for this ton of ore seems to be unusually high, and a cost of \$30 per ton would be nearer the average. The cost of mining will increase considerably with the depth of the workings, especially as it is improbable that any extensive single deposit will be discovered, and the prospecting and development of small deposits are expensive.

As a rule the mine operators keep no detailed account of mining and other costs. Such an account should be kept, in order that the actual profit per ton of ore mined may be arrived at and the efficiency of operations estimated. As a suggestion in the interest of greater efficiency, a short outline of what properly kept accounts should show is given herewith:

The actual cost of mining and sorting per ton of ore.

Cost of powder, fuse, and tools per ton of ore.

Cost of bags, sacking ore, and sewing bags.

Cost of hauling.

Cost of freight.

Pro rata cost of management, charge for depreciation of investment (property and equipment), assayers' charges, ore sampling charges, etc.

In case the ore is milled, the accounts should show the cost of concentration per ton of ore, including the cost of fuel, water, and haulage, as well as charges for amortization of plant, repairs, etc.

The sums expended in establishing a camp and in prospecting for ore should be considered and charged against the tonnage obtained from properties found to be valuable. As an example, let it be assumed that an ore contains 2.5 per cent U_3O_8 and 4.5 per cent V_2O_5 , with a value of \$97 per ton, and that the costs for each ton of marketable ore are as follows: Mining and sorting, \$30; powder, tools, etc., \$2; bags, sacking, etc., \$4; hauling, \$20; freight, \$14.50; then the actual cost per ton is \$70.50. If 200 tons of marketable ore are mined yearly \$10 per ton should be added for management and other charges, making the total charge per ton \$80.50. The net profit is \$16.50 per ton of ore shipped.

If low-grade ores are concentrated, the calculation of cost must include charges for concentration, amortization, etc., at the rate of about one-half ton of shipping ore recovered for each ton of shipping ore hand sorted without concentration, plus the extra expense, if there is any, for mining low-grade ore.

TRANSPORTATION AND PRICES.

The cost of getting the ore to market from the different localities varies considerably. Unfortunately practically all of the ore deposits are miles from a railroad. Some of them are so far away that it is impossible to sell the ore at a profit at the present time.

At Meeker mining operations have not yet reached the stage where ore can be sold at a profit. It costs between \$20 and \$25 per ton to deliver the ore at Rifle. To this must be added freight charges, which are about \$11 to New York.

The cost of transportation from the Skull Creek deposits is prohibitive, except for high-grade ore. It costs \$20 per ton to carry the ore to Vernal, \$15 per ton from Vernal to Watson, and \$8 per ton from Watson to Mack. The freight rate from Mack to New York is about \$12 per ton.

The cost of transportation for the Green River ores is probably less than that for any other of the carnotite deposits. The cost of the haul from the Lorimer camp to the railroad is \$4.50 per ton and the freight rate is \$13 to New York.

The rates at Thompsons are a little higher. Not enough work has been done to be able to give an exact figure, but the haul from the mines to the station would probably be about \$6 per ton and the freight rate practically the same as that from Green River.

The cost of haulage varies widely in Paradox Valley. This difference depends more upon whether pack trains have to be used in addi-

tion to wagons than upon the actual distance of the mines from Placerville. At Long Park very little ore has to be packed and most of it can be loaded directly on the wagons, the cost of hauling to Placerville being \$20 per ton. Saucer Basin has to stand this charge for hauling and in addition a pack rate of \$8 to \$10 per ton. It is somewhat difficult to find out the exact cost of packing the ore on burros, as many of the companies own their own animals, but the figures here given may be considered approximately correct. From the East Paradox mines of the Standard Chemical Co. and the General Vanadium Co. the cost of hauling is \$18 per ton; in addition the ore has to be packed down the mountain side at a cost of probably \$2 or \$3 per ton. From Bull Canyon the cost of hauling is at present \$20 per ton, with a pack rate of \$5 per ton. This figure will probably be reduced slightly after the completion of a road that is being built. The freight rate from Placerville to New York is \$11.57 per ton, and from Placerville to Hamburg or Liverpool, via Galveston, \$14.50 per ton. The average cost of transportation of the ore is, therefore, \$29.57 to \$42.50 per ton. In the case of some of the outlying districts the cost per ton is even more than the higher figure. Adding to this an average cost of \$34 per ton for mining, sorting, and sacking, the total cost to the operator is from \$63.57 to \$76.50 per ton. This does not include costs of powder, tools, assayer's charges, etc.

Up to the present time there has been a ready market for ore containing 2 per cent U_3O_8 or more. Most of the purchasing agents have refused to take any ore of a lower uranium content, although a small quantity of ore has been sold during the past year containing 1.7 or 1.8 per cent. All ore is bought on the basis of its uranium content, a high-grade vanadium ore being difficult to sell if it contains less than 2 per cent U_3O_8 . This is one of the unfortunate features of the present mining conditions, much vanadium ore carrying a little uranium being left on the dump and in the mine. One carload of ore that contained 8 per cent V_2O_5 but only $1\frac{1}{2}$ per cent U_3O_8 was held up indefinitely. If this ore had carried 2 per cent U_3O_8 and less than 3 per cent V_2O_5 , it could have been readily sold. The ore is purchased for the radium that it contains and not for the uranium and vanadium, which are considered as by-products. The foreign buyers established the 2 per cent minimum for uranium and seem to be indifferent to the percentage of vanadium in an ore.

The prices paid for ore vary within narrow limits. One agent offers for 2 per cent U_3O_8 ore \$1.30 per pound of uranium oxide; for $2\frac{1}{2}$ per cent ore, \$1.40; and for 3 per cent ore, \$1.50. For the V_2O_5 content he pays \$0.30 per pound. These prices are f. o. b. New York. An operator who has received offers from several agents states that the prices quoted vary from \$1.25 to \$1.40 per pound of uranium oxide for 2 per cent ore and \$0.35 per pound of vanadium oxide for

ore containing more than 3 per cent of this oxide. Some operators sell their output entirely upon the basis of its uranium content and get nothing for the vanadium. Such a basis usually prevails where the ore is fairly high in uranium and low in vanadium. Where the ore is high in vanadium it is sold for both its vanadium and uranium content according to the prices stated above. When the vanadium is not paid for, the average price given for ore containing 2 per cent U_3O_8 is \$2 per pound and for 3 per cent ore \$2.25 per pound. Little more is offered for ore containing 3 to 5 per cent U_3O_8 , but for ore containing more than 5 per cent the rate is higher. The high-grade material from "bug holes," carrying 12 to 20 per cent U_3O_8 or more, brings about \$3 per pound of the oxide. These prices are all f. o. b. New York or Hamburg.

Deducting the costs of mining and transportation from these prices leaves a very small profit for 2 per cent U_3O_8 ore. One unfortunate feature of the system of marketing is the fact that much ore passes through the hands of four or five agents before it reaches the final purchaser. With better mining conditions and a lower cost of production, the elimination of some of the middlemen, and with prices for ore based on the fact that radium is the main valuable constituent, the operator should be able not only to get a reasonable profit on 2 per cent ore, but also to realize a small profit on ore containing $1\frac{1}{2}$ per cent U_3O_8 , especially if it is high in vanadium.

CONCENTRATION OF ORES.

NECESSITY FOR CONCENTRATION.

In the course of investigations of these uranium and vanadium ore deposits it was found that much low-grade ore was left in the mines, and thus lost, and that there was great loss in hand sorting. Since the market demand is for a material containing at least 2 per cent uranium oxide, the miner, that he may ship no ore below this limit, eliminates all ore that in his opinion contains less. As a result much low-grade and also some shipping ore are left in the mine or are thrown on the dump. By concentrating the low-grade ores at or near the mine these wastes can be reduced. There is no doubt that the ores can be concentrated. Tests made on a small scale substantiate this statement.

In concentration by mechanical means the dry as well as the wet method may be used to advantage; the dry method is perhaps preferable, as little or no water is available in many places where these ores occur. Most of the deposits are far from a railroad, so that wagon hauls are long and expensive. By concentration the bulk of low-grade ore to be hauled can be reduced and at the same time the percentage of mineral in the concentrate raised to a marketable point. Thus a

saving in hauling and freight rates is effected, and the concentrate on account of being richer should bring a better price. Low-grade ores thrown on the dump or left in the mine constitute a natural waste. Much of the low-grade material left in the ground during former operations and perhaps much of the dump material can never be recovered. Much of the ore thrown on the dump disintegrates on exposure to the air, so that much of the valuable contents is washed or blown away. Some of the minerals in the dump rock are liable to be leached out by the rain.

WET CONCENTRATION.

The carnotite ores are of great importance, especially on account of their uranium and radium content. Carnotite, which contains both uranium and vanadium, is a yellow, crystalline, pulverulent material, with a specific gravity of 4.136. It occurs in white sandstone as an incrustation on the faces of joints and fractures and is deposited around and between the individual grains of the sandstone, often strongly adhering to the even surface of a grain. The size of these grains varies greatly, but is generally from a little less than 0.1 to 0.2 mm. in diameter.

In concentrating such an ore the rock should be broken and the pieces reduced to about 40 to 80 mesh with crushers or rolls, the crushing being accompanied by rubbing in order to loosen the carnotite from the sand grains. Further attrition should be applied to the 40-mesh material by convenient means. Fine grinding should be avoided, as it produces too much slime.

If the crushed and rubbed material is washed in a revolving tank or trommel provided with a rubbing device, the particles of carnotite that are in suspension can be drawn off after the grains of silica have settled in the tank or cylinder. As some of the carnotite is carried down by mechanical action with the silica in settling, more water should be added and the operation repeated as many times as may be profitable. With such a treatment a large part of the valuable minerals can be extracted.

The slime is washed into settling tanks or other suitable device, and the water after the settling of the slimes can be used over again, as will be necessary in the arid districts, where most of the deposits are located and where water is scarce. Enough water can be collected and stored away for this purpose from seepage, springs, and during rainy days, and in many cases a wet treatment can in this way be made possible.

RESULTS OF WET CONCENTRATION TESTS.

Tests have been made of ore taken from the dumps at various claims. In one instance this ore contained 2.02 per cent U_3O_8 and 2.32 per cent V_2O_5 . Concentration by the wet method gave the following results:

Results of concentration by wet method.

	Quantity.	V_2O_5 .	Weight of V_2O_5 .	U_3O_8 .	Weight of U_3O_8 .
	Grams.	Per cent.	Grams.	Per cent.	Grams.
Ore.....	1,943	2.32	44.08	2.02	39.348
Concentrate.....	191.3	9.92	18.976	8.84	16.91
Residue.....	1,751				
	1,942.3				

Loss in concentration, 0.7 gram. Concentration ratio=10.156:1. Extraction from ore by concentration, V_2O_5 =43.035 per cent; U_3O_8 =43.08 per cent. Ratio of ore to concentrate, vanadium content=1:4.275; uranium content=1:4.376. Ratio of radioactivity, measured in the electroscope, ore to concentrate=1:4.56, which shows that the concentrate is 4.56 times more radioactive than the ore.

Other tests made with ore containing less U_3O_8 and V_2O_5 have shown that an extraction can be made with a proportionately similar result.

The following result was obtained with a reddish-brown vanadium-bearing sandstone from Utah. The ore was crushed to the size of the grains, rubbed, and then washed in cold water. The slimes thus obtained were settled and decanted or filtered.

Result of concentration of vanadium-bearing sandstone by wet method.

	Quantity.	V_2O_5 .	Content of V_2O_5 .
	Grams.	Per cent.	Grams.
Ore.....	400	5.57	22.28
Concentrate.....	85	17.35	14.75
Residue.....	315		

Concentration ratio=4.72:1. Extraction=66.2 per cent.

According to electroscopic measurements, and by comparison with a standard material of known U_3O_8 content, it was found that this ore contained approximately 0.71 per cent U_3O_8 and the concentrate contained 1.62 per cent.

In concentrating this vanadic sandstone the content of vanadium was therefore raised to 17.35 per cent V_2O_5 and the concentrate can be sold in competition with the Peruvian patronite. In addition the uranium and the radium can be easily extracted from this ore. Some of the vanadium can not be extracted mechanically; it probably exists in the sandstone as roscoelite, which is bound in the silica.

DRY CONCENTRATION.

Where water is scarce, a dry process can be adopted for low-grade ores. As good concentration with a dry as with a wet process can not be expected, but an equally efficient if not a better extraction can be obtained.

In a dry process the rock should be reduced with crushers and rolls as nearly as possible to the size of the grains of the sandstone. Much of the carnotite can be obtained by sifting the crushed material through a 120 to 150 mesh screen. Care must be taken to collect all of the dust, as this is richest.

Good results can be obtained by using an air current to blow the finest particles into a dust chamber, in which they are collected. The air current must be steady and evenly expanded, so that the coarse particles will drop out of it, and strong enough to completely lift the material. The coarse material should be carried along with the current for some distance, so that as much as possible of the powdery material can be freed and blown from the coarse grains, which should finally drop without carrying the powder with them. These particles should fall into a discharge spout and thence into bins or other suitable device. The powdery material, which carries the valuable minerals, is collected in dust chambers. These chambers must be perfectly tight. Much valuable mineral has been lost in sampling shipments when the sampling device had no arrangement for saving the dust.

A number of machines that doubtless could be used for the dry separation of these ores are now on the market.

Another method of dry concentration of low-grade ores consists of simple sifting, with previous rubbing, of the crushed ore. The ore is reduced with crushers and rolls to about 40 mesh and is then passed over an oscillating 150-mesh screen. The undersize from the 150-mesh screen is concentrate. The oversize is carried to a device in which the sand grains are subjected to a thorough rubbing, which removes from the silica the adhering fine particles of carnotite.

In the experiment stiff wire brushes rubbing against a steel plate were used with success for the attrition. The entire material was then brought over two screens, one overlying the other; the upper screen being 80 mesh and the lower one 150 mesh. The table below clearly shows what results can be obtained by such a process. The screens must be encased and all of the dust must be collected. The dust from the sifting operation and that which remains in suspension in the dust-proof screen boxes can be blown from these boxes by suitable means into dust chambers and then added to the concentrate.

A concentration by this dry method on a small scale gave the following results:

Results of dry concentration by sifting with previous rubbing.

	Mesh of sieve.	Quantity.	V ₂ O ₅ content.	Weight of V ₂ O ₅ .	U ₃ O ₈ content.	Weight of U ₃ O ₈ .
		Grams.	Per cent.	Grams.	Per cent.	Grams.
Ore.....	40	500	2.40	12.25	2.24	11.20
Concentrate.....	Under 150	134	4.66	6.24	4.42	5.92
Residue.....	Over 150	169	1.60	2.70	1.53	2.58
Do.....	Over 80	197	1.50	2.95	1.37	2.68
		500		11.89		11.18

Concentration ratio—ore: concentrate—3.73 : 1; extraction V₂O₅=50.85 per cent; extraction U₃O₈=52.85 per cent.

Care must be taken in crushing the ore for either wet or dry concentration that little, if any, of the dustlike material is lost. Dust chambers should be attached to the crushers, or the latter should have air-tight casings and the material should not be removed until the dust has settled.

COST OF CONCENTRATION.

The factors to be considered in a calculation of the cost of concentrating the ore by a wet method are:

1. Cost of concentration:

- (a) Water supply.
- (b) Cutting wood for fuel for power and drying (or cost of gasoline if that is used for fuel).
- (c) Hauling of fuel or gasoline.
- (d) Wages for concentrator men.

2. Amortization of equipment for concentration (pro rata of tonnage of material treated per annum for such equipment and for management).

Two men, at \$3 a day each, should be able to operate a small plant. The cost of water supply should not exceed \$2 a day. The cutting of fuel or cost of gasoline should not be more than \$2 to \$2.50 a day. The hauling of the fuel would probably cost \$1.50 a day.

On an average, from 10 tons of ore mined only 1 ton of ore with a content of over 2 per cent U₃O₈ is obtained by hand sorting. From the other 9 tons, thrown on the dump, about 5 tons of low-grade ore can be separated for concentration. Assume that such low-grade ore contains, on an average, about 1 per cent U₃O₈ and 1.5 per cent V₂O₅. With a concentration ratio of 10 to 1, half a ton of shipping ore should be obtained. The total extraction of mineral by a wet method should be about 50 per cent, and the

grade of the material obtained would therefore be fairly high. In an actual test on such a low-grade ore the percentage of U_3O_8 in the concentrate was found to be about 4.5. This should of course command a proportionately higher price.

The cost of concentration by the dry process is approximately the same, less the cost of the water supply.

The cost of concentration, including ample charges for amortization and for management, should not exceed \$20 per ton of concentrate. These figures are conservative and probably the ores can be concentrated more cheaply.

The average cost of mining with present methods at the rate of 10 tons of ore a day with 6 men and the hand sorting done by the foreman is about \$30 per ton of shipping ore. This figure does not take into account the powder used in blasting, the wear of tools, etc. If the low-grade ore is utilized by means of concentration, the expense of mining is thereby increased little, if any, and only the additional cost for the concentration has to be taken into consideration, the charges for sacking and hauling the concentrate to the station being the same as for a ton of ordinary ore.

From the above it will be seen that through utilizing the low-grade ores the average tonnage per annum could be increased 50 per cent with little extra expense. But as the uranium and vanadium content of the concentrate would be at least double that of the average ores shipped, the total production of uranium and vanadium from these ores would be doubled. For various reasons it is hardly to be expected that all of the waste material can be utilized, and therefore such a large increase will probably never be actually obtained.

Perhaps several of the operators could combine and install an equipment for mechanical concentration and treat their low-grade ores in a central plant, especially where several operators are in the same district and the claims are near each other. With such a plant in use the production of marketable ore at the present rate of mining could be increased by about one-half without increasing the cost of mining and with only the additional expense of concentration. This increase can be obtained by treating material that hitherto has been lost.

The assumption, which has been made by many of the dealers, that by concentration a large part of the radium content of the ore would be lost, is entirely without foundation, as no such losses of radium in the concentrate can occur from any mechanical treatment of the ore. This has been proved by our tests of the concentrates (pp. 37, 39). Undoubtedly the statement has reference to chemical concentration, in which, of course, such losses might occur.

POSSIBILITY OF CHEMICAL TREATMENT.

A chemical treatment of the low-grade carnotite ores at or near the mine would undoubtedly be better and a larger extraction could be obtained were it not that the present high cost of hauling chemicals to the mine prohibits such treatment. In addition, the average operator could not obtain such labor as would be necessary in a chemical process. The deposits worked by the smaller miners are scattered, and their probable output is too uncertain to justify the erection of even a small chemical plant. This statement does not apply to a chemical plant conveniently situated for the treatment of custom ores.

GENERAL STATEMENT.

The possibility of a continuous supply of carnotite ores in the future is naturally of great interest. In the Utah fields, as already stated, the ores are generally low in uranium, although occasional small pockets of high-grade ore are found. Some of these deposits have the advantage of short hauls; others, such as those at Table Mountain, have as long a haul as most of the ores taken out of Paradox Valley. Since the present demand requires an ore containing at least 2 per cent U_3O_8 , the operators in the Utah fields have to do such careful sorting that a large proportion of the uranium ore is wasted. There is therefore, as already explained, great need of a concentration method, or methods, that can be used by an operator at his own camp, or at least by a group of operators working nearby claims. In this way not only would the low-grade uranium ores be made available but also the vanadium ores that contain too little vanadium to be worth marketing under present conditions. It is quite possible that such ores may be so concentrated that the concentrates can be handled for their vanadium content alone, even though they contain too little uranium to warrant their being classified as uranium ores.

In Paradox Valley at present little difficulty is experienced in obtaining 2 per cent ore by careful sorting, and a fair amount of ore carrying 3 per cent or even more has been mined during the past year; but there has been much waste in the sorting of this ore and more low-grade material has been left in the mines untouched. This is particularly true in Saucer Basin and at certain points on the south side of East Paradox Valley. It is difficult to state just how much ore suitable for concentration but not rich enough to ship on the present basis is available in the Paradox and surrounding districts. It is certain, however, that there is enough already in sight to double the output for several years to come, if it could all be made marketable.

As this is hardly possible a 50 per cent increase is probably all that could be expected under the most favorable conditions.

The present policy, which is really necessary on account of the cost of production, of shipping only ore containing 2 or 3 per cent U_3O_8 , is bound to have a far-reaching result. The buyers are urging the shipment of even higher-grade ore and are offering a bonus for it. Hence the tendency has been to raise rather than to lower the minimum percentage, a correspondingly larger waste resulting. Although some of the pockets in Paradox Valley are large, a number of the largest have been worked out within the past two years. The output of 2 per cent ore may be as large for several years to come as it has been during the past year, and may even increase for a while, but after a few years the production of high-grade material will inevitably decline. The buyers will then either have to accept a smaller output or handle low-grade material. New pockets will naturally be opened from time to time and a certain percentage of high-grade material will be available for many years to come, but it is extremely important that methods be devised for utilizing at least some of the low-grade material now being thrown on the dump or left in the mine.

The United States possesses unique deposits in these carnotite ores. They constitute at present the largest known supply of radium-bearing minerals in the world. With the exception of the ore mined and utilized by two firms, practically every pound is shipped abroad. Up to the present very little interest has been shown by Americans in these deposits, which may not be duplicated in so far as quantity goes in any part of the world.

The only other large deposits of uranium-bearing ores known are those in Austria. They are considered of such importance that the Austrian Government has taken entire charge of them. The output from the carnotite fields of this country is much larger than that from the Austrian mines and is likely to continue larger for some time to come, but the ore should be mined with minimum waste and the industry should yield a maximum profit to this country.

There is little danger that the amount of ore mined per annum will increase materially. The long hauls and the scattered occurrence of the deposits make it difficult to largely increase the production in any one year. Such an increase can be accomplished only by utilization of the low-grade ores.

A word of warning may be added at this point. The fact that carnotite and its associated ores are found on a claim does not necessarily mean that such a claim is worth even the assessment work that has been expended on it. Already one company is making extravagant statements in its advertisements, and others may follow it.

PITCHBLENDE DEPOSITS.

DESCRIPTION OF DEPOSITS IN THE UNITED STATES.

Pitchblende has been found in the following localities in the United States: Feldspar quarry, at Middletown, Conn., in large octahedrons; in Hall's quarry, at Glastonbury, Branchville, Conn., in a pegmatite vein and usually embedded in albite; at Marietta, S. C.; in the Baringer Hill district, Llano County, Tex.; in the Bald Mountain district, Black Hills, S. Dak.; in Gilpin County, Colo.; and in Mitchell County, N. C.

NORTH CAROLINA DEPOSITS.

In Mitchell County, N. C., small quantities occur near Penland in the quartz and feldspar mines. The mineral is usually associated with quartz, but in places with orthoclase feldspar, and still more infrequently with albite. Its presence is usually indicated by a dull-green stain on the quartz or spar, although such stains do not invariably mean that ore is present. The pitchblende from near Penland is usually high grade and is often associated with yellow gummite. In probably the best mine only 50 pounds have been found in 1½ years, so that as yet the mining of these deposits can not be considered a commercial enterprise. The ore is usually sold in small quantities as museum specimens at specimen prices.

COLORADO DEPOSITS.

In Gilpin County, near Central City, Colo., are five mines that have produced pitchblende, namely, the Kirk, the Wood, the German, the Belcher, and the Calhoun. These mines are all within about 2 miles of Central City and are situated on or very close to Quartz Hill, at an altitude of 9,500 feet. The Kirk, Belcher, and German mines are close together on the hill; the Wood and Calhoun are in the valley.

These mines were originally gold mines and until recently have been worked mainly for gold. Gilpin County is the oldest mining district in the State of Colorado, and it was here that the first stamp mills and smelters in Colorado were built. The ores near the surface are mainly free-milling; lower down they change to sulphides. The veins are persistent, but the quality of ore often varies considerably within a short distance.

KIRK MINE.

The original workings of the Kirk mine are about 100 feet deep. The majority of these have been filled up. In this manner the

larger part of the 500 feet of the lode to the east of the shaft has been worked out to the above depth, but the 1,000 feet to the west has not been opened except by prospect holes. An old shaft and part of the workings are still open. In this shaft and workings a considerable amount of pitchblende was mined in the early days. The miners did not know at the time what the ore was and it was wasted in an attempted treatment for gold. When the pitchblende was found the gold gave out, and therefore the owner left his shaft and started a new one 185 feet to the west, which is now the main shaft. The next owners knew the value of pitchblende and about 1898 took out several tons of good ore. When further development was unsuccessful, the mine was bought by the present owner. The mine is more than 400 feet deep. The vein strikes almost due east and west and dips about 80° S. The shaft follows the vein. The main levels are at 97, 140, 200, 300, and 400 feet.

The Kirk lode is 3 to 6 feet wide, and seems to be a fissure vein in gneiss and mica schist. It carries gold, silver, and copper, in addition to pitchblende. The best pitchblende was found between the 140-foot level and the 250-foot level on the east side of the shaft. Ore was also found at other places, notably just above the 400-foot level and in this level. The ore shoot seems to pitch from east to west. Except in one place, the ore was everywhere against the country rock on the hanging wall. There were practically no stringers, spurs, or other indications that the vein was near except that the country rock carried some uranium. Much of the ore was exceedingly rich, some assaying 60 to 80 per cent U_3O_8 . At places this rich ore was a foot thick; one single piece was removed that measured 2 feet 8 inches by 1 foot 4 inches by 1 foot.

Reliable data on the production of pitchblende from the Kirk mine before the present ownership could not be obtained. Since the present owner has had the mine, in round numbers about 20 tons of ore with an average content of 35 per cent U_3O_8 , and something over 100 tons of ore with a content of 3 to 4 per cent U_3O_8 have been mined. Most of this ore was produced in 1905-6, and practically all of the high-grade ore was sold abroad.

Therefore at a time when the Austrian mines were apparently the only producers of uranium ore the Kirk mine was sending pitchblende abroad and supplying a large proportion of the radium that was sold on the open market.

During the last few years the Kirk has not been worked regularly; therefore the production of ore has been small.

Near the Kirk are the Jeannette and Hilda claims. Some assessment work has been done on them.

GERMAN AND BELCHER MINES.

The German and the Belcher mines are a short distance north of the Kirk mine. It has been difficult to obtain satisfactory and reliable information concerning the production of uranium from these mines previous to 1911. About three years ago the Belcher mine produced 1,600 pounds of ore carrying 30 per cent U_3O_8 , and shortly afterwards about the same quantity was mined in the German. During this period both mines were run in a rather haphazard manner with inefficient equipment. Last year both mines were taken over by the German and Belcher Mines Co.

The shaft on the German mine is down 600 feet, but is blocked at the 400-foot level. The main levels are at 130, 250, and 400 feet. The vein dips south, but is nearly vertical, like that of the Kirk mine. The Belcher mine is in the same vein as the German mine, several hundred feet to the east. The shaft is down 200 feet, with levels at 120 and 180 feet.

The country rock in these mines is gneiss and mica schist, similar to that in the Kirk. Intrusive andesite and granite are common. Unlike the Kirk, thin stringers of pitchblende are sometimes found that lead to pockets of rich ore. In addition to the pitchblende, the ore contains iron and copper pyrites and lead and zinc sulphides carrying gold and silver. The proportion of pyrites is larger than in the Kirk mine, and the ores carrying small quantities of pitchblende are much richer in sulphides than similar ores from the Kirk. Rickard^a states that at places a "gouge" or clayey salvage mixed with soft pitchblende ore of low grade is found. This is supposed to be the result of crushing of the vein material during a second period of rock movement. As in the Kirk, pitchblende and gold do not occur together.

There is a good showing of pitchblende in the western part of the 130-foot level. The high-grade ore is from 1½ to 2 inches thick, some of it containing over 80 per cent U_3O_8 . Lower-grade material is also found in the eastern part of the 250-foot level, indicating that the ore shoot pitches east. This is exactly opposite to the apparent pitch of the ore in the Kirk mine.

The company is doing development work chiefly at present and is working more than 20 men in two shifts, is using air drills, and has contracted for a considerable amount of diamond drilling.

The total production of pitchblende ore from the two mines from the fall of 1911 to January 1, 1913, has been 240 pounds of high-grade ore containing more than 70 per cent U_3O_8 , 220 pounds of ore containing 20 per cent U_3O_8 , 5 tons of ore carrying 2.6 per cent U_3O_8 , and 1 ton of ore carrying 2 per cent U_3O_8 .

^a Rickard, Forbes, Pitchblende from Quartz Hill, Colo.: Min. Sci. Press, June 7, 1913, p. 851.

THE CALHOUN MINE.

The Calhoun mine, south of the Kirk and German mines, is at present worked for gold only. There are two shafts on this claim, but the east shaft is not used at present. The mine has been a small producer of uranium. During 1912, \$1,300 worth of pitchblende was sold for specimens—the total output during the year. This was nearly all high-grade material and was obtained at a depth of 387 feet. That portion of the mine from which the pitchblende was obtained is closed until a successful process for the concentration of low-grade ores has been developed.

THE WOOD MINE.

The Wood mine at present is being worked under lease by three men, who have had charge about a year. As in the case of the Calhoun, the gold is sought rather than the pitchblende. During March, 1913, a small pocket of pitchblende was struck, from which 400 pounds of high-grade ore were taken. The vein is narrow, being only 9 to 18 inches wide, but at places the ore is rich in gold. In addition, the ore carries lead, copper, silver, and near the west end of the 200-foot level, also zinc. There are two shafts on the property, with levels at 135, 160, and 200 feet. Most of the ore above the 135-foot level has been stoped out. During the past 5 years about 1 ton of medium-grade pitchblende ore has been mined. This was found a little below the 160-foot level, close to where ore has recently been found.

In general these mines are similar as regards the character of the veins, the quality of the pitchblende, and the depth at which the latter is found. A porphyry dike outcrops at several points just west of the pitchblende mines, and the workings of the Calhoun mine extend from the west shaft into this intrusion. The Alps mine, which is close to the Kirk and German, but which has never produced any pitchblende, lies west of this dike, which may have had something to do with localizing the deposition of the pitchblende. The sulphides, which are found in large quantities in these deposits, are doubtless due to secondary mineralization.

From the past showing of these mines, especially of the Kirk mine, one would be justified in classifying the veins at Quartz Hill as being among the important pitchblende deposits of the world.

WASTE OF LOW-GRADE ORE.

Until recently low-grade material has been largely neglected. Nevertheless the future of these mines depends largely on the utilization of this low-grade material. It is quite probable that much of the rejected material is low-grade uranium ore. Some rejected pieces picked up in the levels of the German mine carried 12 per cent U_3O_8 .

If such material as this was discarded, it is more than probable that a large proportion of lower grade ore was thrown out. A composite sample, taken by one of the writers from eight different points in the dump of the old workings at the Kirk mine, assayed 0.8 per cent U_3O_8 . This sample, of course, was not by any means an average of the whole dump, but it shows that low-grade material has been discarded in the past.

CONCENTRATION OF LOW-GRADE PITCHBLENDE ORES.

Attempts at concentrating the low-grade pitchblende ores have been made by various parties with fair results.

The ore should be hand sorted before treatment by mechanical means, to remove as much of the waste rock as possible. After crushing to the desired mesh, the material should be carefully sized and classified. This is essential, as a high concentration with a small loss of valuable minerals can not be obtained with unsized material. The order of the important compounds in the ore, in regard to their specific gravity is uraninite, pyrite, copper pyrites, and quartz. The difference in specific gravity of pyrite and copper pyrite is small, between pyrite and uraninite considerable, and quartz, being the lightest material, can easily be separated. In one concentrating test both a jig and an oscillating table were employed, and, seemingly, a better separation was obtained with the jig than with the oscillating table. The separation of the uraninite from the pyrite and copper pyrites can be accomplished by flotation, but better results can perhaps be obtained by magnetic separation after a slight roasting of the ore.

The pyrite and copper pyrites must be saved, as they may contain gold.

There is no doubt that a high concentration can be accomplished with little loss. Results obtained by the Bureau of Mines, which is working on the problem of concentrating low-grade uranium ores, will be published in another report. It is expected that ore containing as low as 0.5 per cent U_3O_8 can be utilized. This should help to make mining for pitchblende a better commercial venture, especially as there is a good yield of gold and silver to be derived from by-products.

Mechanical concentration of low-grade pitchblende ores has been done in a similar manner in Austria and Germany for several years, but no definite data in regard to methods and results have been obtained.

EUROPEAN PITCHBLENDE DEPOSITS.

A comparison of the Colorado pitchblende deposits with those of Europe may be of interest, although up to the present the former have not received much attention. The important European deposits

are found in Germany and Austria. The ore deposits at St. Joachimsthal, Austria,^a are in mica schist interbedded with lime schist and crystalline limestone. Toward the east and northeast the formation is gneiss. The gneiss was intruded by quartz porphyry subsequent to the deposition of the vein material. In the mica schist are fissures filled with volcanic material which cut the mineralized zone at various points and depths. The veins are usually 6 inches to 2 feet wide, in rare cases widening out to 3 feet. The mode of mineralization varies greatly. The ores are in both stringers and pockets, and contain the following metals: Silver (metallic and as argentite, polybasite, stephanite, tetrahedrite, proustite, pyrargyrite, sternbergite, and other minerals); nickel (nickelin, chloanthite, and millerite); cobalt (as smaltite, bismutosmaltite, and "absolan"); bismuth (as metallic bismuth, bismite, etc.); arsenic (as metallic arsenic, arsenopyrite); and uranium (as uraninite, or pitchblende and other alteration products). Galenite, zincblende, pyrite, marcasite, and copper occur in minor quantities.

The ore beds show that deposition took place in three periods. The cobalt and nickel were deposited first, then the uranium was deposited, and lastly the silver. In some cases the uranium ore is partly replaced by a dark-violet bituminous fluorspar.

Pitchblende is found at Joachimsthal, in Austria, and at Johanngeorgenstadt, Marienberg, Freiberg, and Schneeberg, in Saxony, and Pflibram, in Bohemia, having a similar origin. Dolomite spar is always present, which has generally a white or yellowish-white color, but changes to a peculiar brownish-red hue where pitchblende begins to appear, and is a dirty gray where it is actually in contact with the ore. Deep-blue fluorspar is always present. Copper pyrite is found in small crystals and masses throughout the pitchblende. Pitchblende is also frequently found disseminated in small grains through a part of the mica schist forming a low-grade ore.

The mines at Joachimsthal have been worked since 1517. In 1545 the production of silver ores declined considerably, but since then the deposits have been mined for bismuth and cobalt. During the last 10 years the mines have been worked for uranium. The Edelleut Stollen has been exploited exclusively for uranium ores and the Austrian Government has erected a factory at Joachimsthal for the handling of these ores.

In the vicinity of Annaberg, on the Saxony side of the Erzgebirge, the silver-cobalt veins resemble those at Joachimsthal. A large number of these veins are in the mountains close to the town. Tin, lead, and copper pyrite are among the minerals which have been mined in the older formations. The more recent deposits contain silver and cobalt, or iron and manganese. The gangue is composed of barite,

^a Beck, Richard, *Lehre von den Erzlagerstaetten*: vol. 1, 1909, pp. 408-410.

fluorspar, quartz, dolomite, and pyrites. There is also some amethyst, calcite, aragonite, kaolinite, and gypsum. The ores are cobalt, silver, and bismuth, with some copper pyrites, galenite, pitchblende, gummite, uranochalcite, and arsenic. There are similar deposits at Johanngeorgenstadt, Saxony. The mountain sides are filled with a network of veins. These veins contain tin and silver-cobalt ores and their strike varies greatly. Where dolomite spar is found, the silver-cobalt ores contain pitchblende, as at Annaberg. In the Gottessegen mine the pitchblende occurs in the spar in pieces 2 to 3 inches in diameter. These mines are worked principally for bismuth ochre, but also for cobalt and nickel.

In the quartzic cobalt-bismuth mines of Schneeberg, Saxony, are found bismutite and various minerals of nickel, silver, and arsenic. There is also some pitchblende, uranochalcite, troegerite, walpurgite, zeunerite, uranospinite, galenite, zincblende, and some copper compounds..

URANIUM DEPOSITS IN PORTUGAL.

The following description of the uranium deposits in Portugal is given by Segaud and Humery: ^a

The uranium-bearing zone lies in the area of massive granite that occupies nearly the entire northern part of Portugal between the Desert of Galice to Castello Branco and reaches into the provinces of Minho, Tras-os-Montes, and Beira. The richest part of the district is between the towns of Guarda and Sabugal, near the southern part of the granite area. The zone touches the outcrops of Cambrian rocks north of Guarda, the veins being especially rich in wolframite. The region of Villar-Formosa, however, is almost equally uraniferous.

This zone appears also south of Guarda, where the granite forms a superficial mass thrown above the primary rocks by faulting. Apart from the uranium, the rocks of the region are much mineralized, showing deposits of galena, arsenopyrite, chalcopyrite, tungsten, and cassiterite. The uranium-bearing veins crop out in the granite, and also in the Cambrian schists. The granite is generally hard, firm, and little altered. In the vicinity of the veins it is altered, perhaps through mechanical action. The degree of alteration varies much at different points. In some places a remarkable parallelism of the veins in one or two directions is observed; in one instance the veins can be followed 8 miles, and in another, less important, for a considerably longer distance. The width of the veins varies greatly. Outcrops show a continuous succession of swellings and constrictions, and the vein in many places disappears completely, only to reappear several yards farther away. In the north of Belmonte a vein is 8 meters wide for a short distance, but such width is believed to be exceptional. A width of from $\frac{1}{2}$ to 1 meter is more common. The filling of the veins is mostly pegmatite and aggregations of crystallized feldspar and quartz. In the district of Guarda the filling of several veins is largely quartz. The gouge is argillaceous matter, and may carry uranium. Autunite (uranium-calcium phosphate) is present in different forms, in small groups of square tables of an intense yellow color, in small plates, as a pure coating, or still oftener in bright yellow specks disseminated through-

^a Segaud, —, and Humery, —, Deposits of uranium in Portugal: Ann. des Mines, ser. 11, vol. 3, February, 1913, pp. 111-119.

out rock of a dull yellowish color. In the clay part of the veins the uraninite is in places completely invisible and its presence is revealed only by the electroscope. There are, also, blotches on the surface of the granite, giving it an intense yellow color. Notwithstanding this color, the granite is always poor in uranium. The autunite is probably accompanied by uranyl-circite. Chalcolite of a beautiful emerald-green color is also present. The content of the veins varies enormously. At one place on an outcrop material was found carrying values of 4 to 5 per cent,^a whereas the surrounding material carried no values whatever. Ore containing 2 per cent is excellent and that carrying 1 per cent is good average ore. The minimum limit for profitable exploitation is a content of 0.3 per cent to 0.5 per cent ore. The unaltered mineral has probably not yet been reached, except perhaps in a few places, and it is presumed that the mineral does not differ greatly from that in outcrops.

The region of Guarda forms an elevated plateau having an altitude of about 2,700 feet. It is traversed by a railway from Pampilhosa to Villar-Formosa and from Guarda to Lisbon. The transportation to the railroad stations is by oxcarts, but the erection of an aerial tramway is possible. The main work has been done in the more important districts of the northeastern part of the region around Porto. The region contains numerous small hamlets from which laborers can be easily procured at small wages.

The most important workings are those of the Compagnie de l'Urane, which employs 600 men. The average net cost per yard of drifts with a cross section of $3\frac{1}{2}$ by $4\frac{1}{2}$ feet is about \$17, and the average net cost of 1 cubic meter of broken ore is about \$20. The small mineral content of the ore makes treatment at the mine necessary. This treatment is entirely chemical, without previous mechanical concentration; the products are uranium oxide on one hand and sulphate of barium, rich in radium, on the other. The most important factory is at Baraao, and is owned by the Compagnie de l'Urane. The radioactive barium sulphate is treated in the factories of Nogent.

URANIUM ORES IN AUSTRALIA.

Within the last few years several finds of uranium-bearing ores have been reported from Australia. These reports, apparently, have interested not only scientific men in Australia, but also the general public. One of these deposits is 80 miles east of Farina, a railroad station on the Great Northern line in South Australia, and lies between Mount Painter and Mount Pitt. Brown^b states that the rocks of the district consist of coarse and fine feldspathic, siliceous, and micaceous granite, and gneiss, and also micaceous rocks, quartzite, and mica schist. The rocks are contorted in places and penetrated by dikes of coarse, pink colored, eruptive granite.

Two of the prospect pits are on outcrops of iron oxide with cellular quartz and gossan, the whole having the appearance of an irregular lode. The uranium minerals, torbernite and autunite, are disseminated through the ore, and are also crystallized on the walls of the fissures and cavities in it. Uranophane and gummite occur sparingly. Fergusonite and some monazite are also present. Torbernite occurs on two of the other claims, and on still another both torbernite and autunite are disseminated through the rock and in seams. The extent and width of the ore deposit have not been determined.

^a The article does not state what these percentages indicate. They probably refer to U_3O_8 .

^b Brown, H. L. Y., *Uranium ores in South Australia*, 1911, p. 3.

Another uranium deposit lies southeast of the one just described, about 20 miles east-southeast of Olary, on the railroad line from Petersburg to Broken Hill, South Australia. The ore occurs as a yellow and greenish-yellow incrustation and powder on the faces, joints, and cavities of a lode, which consists of titaniferous magnetite, magnetite, etc., and quartz in association with black mica. There are two of these lode outcrops, more or less parallel and 5 to 15 yards apart. The main outcrop can be traced for some 200 yards. The ore is reported to consist almost entirely of carnotite, with possibly some gummite.

From the accounts published most of the ore is of a very low grade. A company has been formed for exploiting the ore and extracting the radium.

VANADIUM FROM ORES OTHER THAN CARNOTITE.

SAN MIGUEL COUNTY, COLO., DEPOSITS.

Probably the largest deposits of vanadium that have yet been discovered in the United States are in southwestern Colorado between and close to Placerville and Newmire in San Miguel County. These deposits were visited by Ransome and Spencer in 1899 and their description, together with notes on the chemical analyses and composition of roscoelite by Hillebrand, was published in 1900.^a Fleck and French have also described the deposits.^b Fleck and Haldane later published additional descriptions, with notes on mining operations.^c Hess, in 1912, published an excellent description of these deposits with notes on the possible origin, etc.^d

According to Cross and Purington,^e the country rock is composed of Jurassic and Triassic sediments. They have divided these into three formations, the Dolores below, La Plata above, and McElmo above the La Plata. The latter is composed of two heavy beds of light-colored sandstone, separated by a thin bed of limestone. The vanadium-bearing rock is the lower sandstone which probably consists of two beds with an unconformity between. That part above the unconformity is yellowish in color and that below is gray.

These beds outcrop on both sides of Bear Creek, south of Newmire, at a height of about 50 feet above the creek bed. A chemical company with works at Newmire, Colo., and Primos, Pa., is mining these Bear Creek deposits. On the east side of the creek the workings are rather extensive. The ore is removed as found, although such

^a Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado: *Am. Jour. Sci.*, 4th series, 1900, vol. 10, pp. 124-144; U. S. Geol. Survey Bull. 262, 1905, pp. 10-14, 18-22.

^b Fleck, H., and French, S. W., Uranium and Vanadium: *Quart. Colorado School of Mines*, January, 1909.

^c Fleck, H., and Haldane, W. G., Preliminary report on the radioactivity of carnotite in southwestern Colorado: *Quart. Colorado School of Mines*, October, 1909.

^d Hess, F. L., Notes on the vanadium deposits near Placerville, Colo.: U. S. Geol. Survey Bull. 530, pt. 1, 1911, pp. 142-157.

^e Telluride Folio, No. 57, Geol. Atlas, U. S., U. S. Geol. Survey, 1899, p. 52.

operations have resulted in the formation of large chambers. The mining is easy, as the sandstone roof is hard and easily supported. Beyond these chambers are several drifts. Mining at present is confined to the west side of the creek. There is an easy haul of about $2\frac{1}{2}$ miles from the workings to the plant, with a down grade all the way.

The vanadium-bearing rock is a light to dull green fine-grained sandstone. Sometimes the color is quite dark. Occasionally splotches of carnotite are found in the cracks and fissures, but the uranium content is too small to be worth saving. As a rule the richest ore follows the apparent unconformity between the two sandstones. In places a shallow layer, dark in color and from one-half to one inch thick, lies close to the unconformity and is said to be rich in vanadium. This layer partly, at least, accounts for the origin of the vanadium in the sandstone above and below.

The deposits are also found at Sawpit between Newmire and Placerville, on both sides of the Rio Grande River; in the canyon of Fall Creek which runs into the Rio Grande below Sawpit; on the east side of the Rio Grande both north and south of Placerville; and on both sides of Leopard Creek, which runs into the Rio Grande close to Placerville. The only ore being mined at present, however, is at Bear Creek. According to Hillebrand,^a the green vanadium mineral to which the sandstone owes its color is not a chlorite notwithstanding its appearance, but is closely related to the mica roscoelite wherein the proportions of Al_2O_3 and V_2O_5 are reversed. A large proportion of the ore carries less than 1 per cent V_2O_5 . The ore mined at Bear Creek has an average content of $1\frac{1}{2}$ per cent V_2O_5 . Some of it contains as much as 2 per cent V_2O_5 , or even a little more. The thin layer, already referred to, at places carries more than 8 per cent V_2O_5 . These low-grade roscoelite deposits can be mined at a profit, because they are large and easily worked.

HUERFANO COUNTY, COLO., DEPOSITS.

Vanadium ore has been recently discovered in Huerfano County, Colo. The Colorado Mining Corporation claims, six in number, are in the Culebra spur of the Sangre de Cristo Range in Huerfano County. The nearest railroad station is Russell Siding, about 9 miles distant. From this siding to within a short distance of the mines the wagon road is an abandoned railroad grade. The vein is said to be a well-defined fissure vein and has been opened up at different places on the surface for a distance of 2,500 to 3,000 feet. It is 1 to 4 feet in width. The lowest workings are about 25 feet deep with the best showing in a couple of shafts about 14 feet in depth. The vein was originally

^a Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado: U. S. Geol. Survey Bull. 262, 1905, p. 21.

worked for copper, but about a year ago it was found to contain vanadium. A number of assays show the following V_2O_5 content: 2.63, 2.91, 6.25, and 7.35 per cent. The copper content is 2 to 4 per cent or a little more. One assay made upon a much larger sample than was used in any of the above determinations showed 4.5 per cent V_2O_5 . The ore is heavy, black, and banded and is probably gneiss impregnated with vanadium minerals. It carries much green material, some of which may be copper vanadate. In places the ore is rather yellow and assays are reported showing as much as 1.75 per cent U_3O_8 , but none of the yellow samples we examined contained more than a trace of uranium. Two or three small outcrops carrying vanadium have been found within a mile and a half east of these claims. Five miles north is a thin vein from 1 to 2 inches thick called the Santa Rosa vein, which also is said to carry some vanadium.

CUTTER, N. MEX., DEPOSITS.

At Cutter in Sierra County, N. Mex., on the Atchison, Topeka & Santa Fe Railroad, deposits of vanadinite have been worked by the Vanadium Mines Co. The veins lie about three-fourths of a mile south of Palomas Gap and contain, in addition to vanadium, galenite, copper carbonates, barite, fluorite, and other minerals. They were visited by Hess in 1911.^a The mine has recently been abandoned and the plant removed.

Some ore has been mined on the Widner claims during the past year, but it was shipped for the lead content only.

EAGLE COUNTY, COLO., DEPOSITS.

In Eagle County, Colo., 7 miles southeast of the town of Eagle, silver ore has been found that carries also vanadium. The Lady Belle mine is 400 feet above Brush Creek on the side of Horse Mountain. There are two tunnels, one about 25 feet above the other. The upper one runs about north and follows the strike of the ore body, the dip of which is 34° E. The lower tunnel runs at an angle to the upper one and then bends until it takes almost the same direction. It is in about 60 feet and the upper one about 40 feet. The ore is a dark-greenish sandstone similar in appearance to the darker types of roscoelite found at Newmire, Colo. It assays 25 to 1,000 ounces of silver to the ton. Much of the ore that is high in vanadium is low in silver, although this is not invariably true. The vanadium ore contains coarsely crystalline layers. The high-grade silver ore is more compact, usually darker, and has blue-black dots and splotches, due probably to silver bromide. Pieces of float ore rich in silver have been found near Horse Mountain, but no high-grade ore has

^a Hess, F. L., Notes on the vanadium deposits near Placerville, Colo.: U. S. Geol. Survey Bull. 530, pt. 1, 1911, pp. 157-160.

been found in place at the time of writing, except at the Lady Belle mine.

A section across the ore body, beginning with the foot wall, is as follows: Four feet of rich ore (A), 11 feet of low-grade ore (B), 4 feet of rich ore (C), and — feet of low-grade ore (D). The hanging wall has not yet been reached, and therefore the total thickness of the ore body is not known.

Vanadium assays made by the Bureau of Mines show the following results:

Vanadium content of ore from Lady Belle mine.

Part of section of ore body.	V_2O_5 content. Per cent.
A (lower part).....	2.66
A (upper part)	.21
B.....	2.30
C.....	1.02
D.....	.38

A sample across the breast of the lower drift showed 0.94 per cent of V_2O_5 . No attempt to recover the vanadium has been made.

DEPOSITS IN OTHER STATES.

A deposit of vanadium ore has recently been discovered in California. It is on low ground, a few rods from a good road, 5 miles from Klinefelter Station, on the main line of the Santa Fe Railroad, near the eastern border of San Bernardino County. It is stated that the vein is 8 feet wide and can be traced on the surface for a distance of 435 feet. The ore is largely calcite. One sample tested by the Bureau of Mines showed a content of 1.71 per cent V_2O_5 . Water is near by, there being numerous springs.

There are several deposits of grahamite in the United States, those in West Virginia, Oklahoma, and Nevada probably being the most important. Grahamite is a solid native bitumen, the origin of which was first described by White^a as being derived from the oxidation of petroleum. Richardson^b mentions the fact that both the West Virginia and Oklahoma grahamites contain vanadium in the ash. The mineral has been described as a brittle, solid, native bitumen, the result of the metamorphism of petroleum, generally pure, but at times containing adventitious mineral matter. Grahamite does not melt, but glows and burns slowly on the application of heat.

The deposits near Page, Le Flore County, Okla., and in the Impson Valley, are fully described by Taff.^c At Page very little development work has been done. An adit has been driven in the vein and within the adit a shaft has been sunk in the deposit, following the

^a White, I. C., Origin of grahamite: Bull. Geol. Soc. of America, 1899, vol. 10, pp. 277-284.

^b Richardson, Clifford, Grahamite, a solid native bitumen: Jour. Am. Chem. Soc., vol. 32, pt. 2, 1910, pp. 1033-1049.

^c Taff, J. A., Grahamite deposits of southeastern Oklahoma: U. S. Geol. Survey Bull. 380, 1908, pp. 286-297.

vein for a considerable depth. The ore outcrops several hundred feet higher up on the hillside. The property has been leased to a Pittsburgh concern for the last few years, but the lease does not call for the working of the property and it has been idle during this time.

The Oklahoma grahamite burns with a smoky flame to a yellowish-brown ash, whereas the grahamite from West Virginia, under similar conditions, forms a pasty asphaltic mass when the volatile matter is driven off, and finally reduces to a reddish-brown ash.

Samples analyzed by the Bureau of Mines laboratory at Denver, Colo., gave the following results:

Analysis of grahamite for vanadium.

	Grahamite from West Virginia.	Grahamite from Le Flore County, Okla.	
		From drift.	From outcrop.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Ash	2.11	0.83	3.39
V ₂ O ₅ content in ore	.17	.28	.22
V ₂ O ₅ content in ash	7.94	34.5	6.77

DEPOSITS OF PATRONITE IN PERU.

The vanadium ores from the mines in the United States meet a strong competitor in the patronite^a shipped from the deposits in Peru owned by the American Vanadium Co., of Pittsburgh, Pa.

These deposits are at Minasragra, 20 miles from Cerro de Pasco. The area lies along the western limit of a broad anticline in "Juratrias" and Cretaceous rocks. A section shows the series in this locality to be composed of green shales, thin beds of limestone, and red shales. Vanadium is found only in the red shales. The deposit proper appears to be a lens-shaped mass, 28 feet wide and 350 feet long. The dip is 75° W., and the strike is N. 20° W. The ore contains several minerals. The mineral that constitutes the larger portion of the deposit has been called "quisqueite." It is a black carbonaceous substance containing sulphur, with a hardness of 2.5 and a specific gravity of 1.75. There is also a lesser quantity of a cokelike material with a hardness of 4.5 and a specific gravity of 2.2. Neither of these contain vanadium. The vanadium is mostly at the southern end of the ore body, and to a depth of 20 feet is largely in the form of red calcium vanadate. The color is brighter than that of the calcium vanadate found in Colorado and Utah, and the ore carries as much as 50 per cent vanadium oxide. It occurs in small

^a Hewett, D. F., A new occurrence of vanadium in Peru: Eng. and Min. Jour., vol. 82, Sept. 1, 1906, p. 385; Hillebrand, W. F., The vanadium sulphide, patronite, and its numerous associates from Minasragra. Peru: Jour. Am. Chem. Soc., vol. 29, pt. 2, 1907, pp. 1019-1029; Bravo, J. J., El Vanadio de Minasragra: Inform. Mem. Soc. Eng. Lima, 1906, pp. 171-185; Hewett, D. F., Vanadium deposits in Peru: Trans. Am. Inst. Min. Eng., vol. 40, 1909, p. 291.

pockets and fills the cracks and fissures in a fine shale. Below this shale is the "mother lode." It is 9 to 30 feet thick, extends along the greater length of the deposit, and dips 40°. It carries as high as 10 per cent vanadium oxide and nearly as much sulphur. On the east and south sides, below the "mother lode," is found a hard blue-black vanadium shale, carrying as much as 13 per cent vanadium oxide and 4 to 5 per cent sulphur. Patronite, the main vanadium mineral, is greenish black and has a hardness of 2.5 and a specific gravity of 2.71. It contains from 19 to 24.8 per cent vanadium oxide and sometimes 50 to 55 per cent of combined sulphur. The patronite originally almost reached the surface close to a dike on the east side and the vein was followed in sinking the shaft. It is most abundant in the north half of the lens. The whole ore body is almost completely inclosed by porphyry dikes. There are also two or three intrusions in the ore body.

PRODUCTION OF URANIUM, VANADIUM, AND RADIUM.

During the year 1912, 28.8 tons of uranium oxide, equivalent to 24.4 tons of metallic uranium, was produced in this country. In addition, 1.4 tons of uranium oxide was shipped, but has been held up in transit because the uranium oxide content was so low that it could not be marketed. The value of the uranium content of the ore shipped, on a basis of \$1.50 per pound, is \$86,000. On the basis that the radium is in equilibrium (see p. 66) with the uranium, 9.77 grams of radium chloride, or 12.7 grams radium bromide (anhydrous), were contained in the uranium ores mined and shipped in this country during 1912. All but a few tons of these ores, as already stated, was sent abroad and the radium was extracted in Europe. It is difficult to say at this time what the exact ratio of radium to uranium in carnotite ore is, as very little work has been done on the subject except in one commercial laboratory. Private information from this laboratory indicates that the ratio is only a little below the normal ratio of 1 part radium to 2,940,000 parts uranium. Allowing a generous margin of 10 per cent, the actual production of radium from American ores last year, assuming that all was extracted, was equivalent to 8.8 grams of radium chloride, or 11.43 grams of the bromide, worth, at the present price of \$90,000 per gram for chloride, \$792,000.^a

We have been unable to ascertain the exact production of radium from other sources during the year 1912, as the figures from the Austrian mines are not yet available. The total production of radium preparations from the Austrian mines in 1911, calculated as

^a Some of the ore shipped toward the end of 1912 did not reach Europe until after the close of the year, and the radium in it would probably not be extracted from it for several months. Although the production of carnotite ore in the United States in 1911 was only a little less than that of 1912, less of it was sent abroad and what was shipped at the end of 1911 would not quite offset what was shipped at the end of 1912, so the above figures on production of radium in 1912 are a little high, although it represents the radium in the ore shipped.

pure radium chloride, was 2.647 grams, valued at \$211,750. The production of radium from other uranium ores mined in 1912, omitting Austria and the United States, is probably less than $1\frac{1}{2}$ grams of radium chloride. The total production of radium chloride from foreign ores, therefore, during the year 1912, assuming that the Austrian production was no larger in that year than in 1911, was less than 4 grams. Therefore American ores supplied more than twice as much radium as was obtained from all other sources.

American ores in 1912 supplied approximately 285 tons of vanadium metal in the form of ferrovanadium and other vanadium products. Some of this went abroad, but most of it was used in this country.

USES OF VANADIUM, URANIUM, AND RADIUM.

USES OF VANADIUM.

The main use of vanadium is as an alloy in steels where great toughness and torsional strength are required, such as automobile parts, gears, piston rods, tubes, boiler plates, tires, transmission shafts, bolts, gun barrels, gun shields, and forgings of any kind which have to withstand heavy wear and tear. The vanadium content in such steels varies from 0.1 to 0.4 per cent. It is occasionally used in certain tungsten alloys for making high-speed tool steel. The introduction of a small proportion of vanadium decidedly reduces the proportion of tungsten required to give such alloys the desired hardness and toughness.

Arnold ^a has given some illustrations of the effect of vanadium upon steels of different types:

One plain carbon steel containing about 1 per cent of carbon had a yield point of 35 tons per square inch, a maximum stress of 60 tons per square inch, an elongation of 10 per cent on 2 inches, and a reduction of area of 10 per cent. The addition to this steel of about 0.6 per cent of vanadium raised the yield point from 35 to 65 tons, the maximum stress from 60 to 86 tons per square inch, still leaving an elongation of 7 per cent and a reduction of area of 8 per cent.

A steel containing 0.25 per cent of carbon and 3.3 per cent of nickel gave a yield point of 33 tons, a maximum stress of 42 tons per square inch, an elongation of 26 per cent on 2 inches, and a reduction of area of 53 per cent. A practically identical steel, but containing in addition about 0.25 per cent of vanadium, gave a yield point of 50 tons instead of 33, a maximum stress of 68 instead of 42 tons per square inch. The elongation was 17 per cent on 2 inches and the reduction of area 36 per cent.

A steel containing 0.25 per cent of carbon and about 1 per cent of chromium registered a yield point of 27 tons and a maximum stress of 41 tons per square inch, with an elongation of 36 per cent on 2 inches and a reduction of area of 55 per cent. The addition of 0.25 per cent of vanadium raised the yield point from 27 to 40 and the maximum stress from 41 to 55 tons per square inch. The elongation was lowered from 36 to 26 per cent and the reduction of area from 55 to 53 per cent.

^a Arnold, J. O., Some recent advances in scientific steel metallurgy: *Nature*, March 20, 1913, p. 70.

Vanadium, therefore, differs from tungsten in having an extremely beneficial effect not only on tool but also on structural steel. Arnold has shown that vanadium seemingly does not form a double carbide with iron, but gradually takes the carbon from the carbide of iron until, if about 5 per cent of vanadium is present, Fe_3C can not exist, and only a vanadium carbide, V_4C_3 , containing 15 per cent of carbon, is present, and this constituent is constant, at least in tool steels containing 5 to 14 per cent of vanadium. The micrographic analysis of such alloys has resulted in the discovery of three new constituents, namely, vanadium pearlite, vanadium hardenite, and vanadium cementite.

There seems to be a tendency to substitute the use of titanium to some extent for that of vanadium, although titanium probably acts only as a reducing agent. Vanadium is also used in making bronzes, in medicine, and in dyeing.

USES OF URANIUM.

Uranium salts have been used for many years in glass manufacturing. Uranium colors glass yellow, and in sufficient proportion imparts to glass a beautiful fluorescent color known as "opalescent." Fifteen per cent or more of the oxide may be required to give the desired effect. It is also used in ceramics for the purpose of obtaining brilliant, fireproof tints of yellow, orange, and black. Uranium coloring powders may be obtained in black or in six shades of yellow.

Uranium can be used as an alloy of steel, but alloys of other metals that have similar properties can be produced more cheaply. Owing to the increased supply of uranium, however, experiments are once more being tried with the object of getting some alloy with properties of a sufficiently distinctive character to make it a commercial product.

USES OF RADIUM.

Radium is used in scientific research and in medicine. A study of radium and its disintegration products has vastly extended the conception of the composition of matter and the nature of the elements. Owing to the cost of the material, however, the quantity available for scientific research must necessarily be limited, although it is unfortunate that a larger proportion of the radium supply can not be devoted to purely scientific purposes. The commercial demand for radium must depend largely upon what use can be made of it for medical purposes. The following data have been abstracted from scientific journals and in part obtained from the bulletin published by the Imperial Department of Public Works in Austria:

Radium treatment is given by means of baths in radioactive waters, by drinking radioactive waters, by subjecting the patients to the radium emanation, by doses of

the radium emanation artificially dissolved in water to a much greater strength than can be obtained from the natural waters, by subcutaneous injections of water containing the emanation in solution, and by direct exposure to the radium rays from radium preparations.

A great number of natural springs have been found to be radioactive. In fact, practically all natural waters contain at least small quantities of the radium emanation in solution, although comparatively few contain radium salts in solution. To say that a water is radioactive, therefore, means very little, as any deep-well water is to some extent radioactive, and even river waters are often slightly so. The important point is not that the water is radioactive, but to what degree it is radioactive. The variation in radioactivity is marked, some springs being hundreds and even thousands of times as radioactive as others. The radioactivity is, of course, acquired by underground water coming in direct contact either with radium emanation which the water dissolves or with radium-bearing ores. Many well-known mineral springs are strongly radioactive, and there has been a tendency to attribute at least some of their therapeutic value to this radioactivity.

Recently the Austrian Government has established baths in connection with the mines at St. Joachimstahl. It is claimed that the waters from the mines have a special value in the treatment of subacute, and especially of chronic, rheumatism of the joints and muscles, for gout, diabetes, and various forms of neuralgia of a rheumatic or gouty nature. It is also claimed that the waters are beneficial in the cases of chronic catarrh of the mucous membranes, slight paralysis, anemia, chlorosis, lymphatic disorders, and diseases of women. The treatment is not confined to baths only, but all of the methods indicated above are used. Direct exposure to radium rays is especially recommended for rheumatism, neuralgia, stiffness of the bones and joints, as well as chronic eczema.

An abstract of part of the recent report of the Radium Institute of London follows:

It is necessary to get a constant source of radiation in the treatment of diseases by means of radium. The different forms of apparatus used for this purpose are called "applicators," and may be of two kinds (a) those containing radium salts, and (b) those containing the emanation or gas evolved by radium salts. The first may be divided into two types, (1) in which the radium salts are fixed by a varnish, and (2) tubes containing the radium salts.

In order to standardize these instruments a unit of concentration has been adopted. It is represented by 1 centigram of radium bromide covering 1 square centimeter of surface. Such applicators are termed "full strength." One with half this quantity on the same area is termed "half strength," and so on. In order to compare instruments with applicators used elsewhere, in future the instruments will be standardized in terms of metallic radium rather than radium bromide.

The varnished applicators are made in the form of square, rectangular, and circular plates of silver. Radium sulphate is mixed with varnish and spread over the metal and, after drying, three individual coats of varnish are laid on top. The whole is finished with a last coat of a particular type of varnish which gives a glasslike surface. Such instruments can be sterilized by means of heating to 120° C., or by means of alcohol, mercuric chloride, or ether. It is recommended that the tube applicators be packed tightly so that the radium salts can not be shaken around inside. The emanation applicators are either flat or in the form of tubes. The flat instruments are small, hollow boxes made of German silver with one face turned down in a lathe to a thickness of 3 mm. The instrument is fitted with a lead tube with a capillary bore. The mixture of oxygen and hydrogen which is obtained from solutions containing radium salts is pumped off by means of a vacuum pump with the contained emanation. On exploding the mixed gases a small quantity of hydrogen is left containing all of

the original emanation. The applicator is then partly exhausted, the hydrogen and the emanation allowed to run in and the end of the capillary tube pinched, conveniently retaining the gases inside.

The tube applicators have a volume of 2 c. c. or more, and are filled with hydrogen containing emanation obtained in the manner already described. If smaller tubes are desired, the emanation is frozen out by means of liquid air and the hydrogen pumped off. The tube can then be sealed.

The institute has also prepared a large quantity of radioactive water for its use, such water having a strength of from 1 to 2 millicuries^a per liter, which is much stronger than even the more radioactive natural waters.

From August 14, 1911, to December 31, 1912, a large number of patients were treated. There seems to have been no disposition to select cases in any way. The general report on these cases is as follows:

Report on patients treated from Aug. 14, 1911, to Dec. 31, 1912.

Examined but not treated.....	38
Received prophylactic irradiation only.....	39
Apparently cured.....	53
Cured.....	28
Improved.....	245
Not improved.....	70
Abandoned treatment.....	38
Dead.....	55
Recently treated and results not yet noted.....	41

These cases covered a large number of diseases, including carcinomata of the larynx, uterus, neck, etc., rodent ulcer, malignant diseases of skin, parotid tumor, adenoma of breast, adenoma of thyroid, fibroid diseases of uterus, leucoplakia, tropical ulcer, various skin diseases, locomotor ataxia, diabetes mellitus, etc.

Statements from another source indicate that 80 per cent of cases of arthritis treated with comparatively strong doses of the radium emanation dissolved in water are either cured or greatly benefited. A similar treatment seems to be beneficial for rheumatism. The dosage in such cases is as follows: The radium emanation obtained in one week from 1 gram of radium bromide in solution is dissolved in 17 liters of water. One hundred cubic centimeters of this solution is taken twice a week by the patient. This means that 1 gram of radium bromide can be used for treating about 170 patients at one time.

A word of warning seems not out of place here. There has already been a disposition to exploit so-called radioactive waters whose radioactivity is no larger than that of ordinary deep-well water. It is by no means an assured fact that even those waters which have a radioactivity somewhat above the average can claim any special healing qualities from their small radium content. It is open to doubt

^aOne millicurie is the quantity of emanation in equilibrium with 1 mg. of radium.

whether cures that have been effected are due to the small proportion of radium, or to other substances in the water. A spring at Seligman, Mo., somewhat largely advertised, was tested by a well-known chemist and proved to have a radioactivity of 3.5 mache units—about the same as deep-well water. The exploitation of radium in medicine has been carried even further. Inhalers, which usually involve the inhaling of oxygen also, and “bath salts,” containing small quantities of radium, are on the market, and the number is likely to increase. It is quite possible that some of these may have a use, but their value has not by any means been thoroughly demonstrated. It must be borne in mind that proper treatment, involving the use of radium, can not be given at present by the average physician, at least to the best advantage, because he is not well informed as to the properties of the substances used. For example, the radium emanation loses half its strength in 3.8 days. That is to say, after 3.8 days half of the gas has changed into disintegration products. At the end of one month all of it has changed and there is no emanation left. Without some knowledge of radioactivity, a physician would frequently not be able to tell the strength of the material he was using.

Although the medical value of radium has been under test for several years, the quantity available for such experiments has been limited. Now that the supply of radium salts has increased a greater proportion of this supply ought to be devoted to scientific research in order that knowledge in regard to its uses in the arts and its medicinal properties may be extended.

RADIUM INSTITUTES.

As already stated, only a comparatively small part of the total supply of radium is at present available for use in medical and other scientific research.

The Radium Institute of Vienna was the donation of Dr. Kuppelweiser, a philanthropist of Vienna, to the Academy of Sciences of Austria, which has turned over the direction of the institute to the department of physics of the University of Vienna. The institute is purely a research institution, offering no courses of instruction and accepting only investigators of recognized standing. It owns about 2½ grams of relatively pure radium salts presented to it by the Austrian Government. About 1 gram of this may be regarded as 100 per cent pure. Although the institute has neither the right to sell nor loan radium, it is the repository for such preparations as the Government is holding for sale and can use them as a source of radiation for experimental purposes. Sales of radium by the Government are made on the basis of the measure of radioactivity of the substance as determined by the institute. The institute has confined its research

work to purely scientific investigations and has rather systematically avoided connection with medical researches.

In Paris the laboratory of Madam Curie is a part of the Sorbonne, or University of Paris, although not at present in the Sorbonne building. It will soon be moved to the new Radium Institute which the university has built near the Pantheon. The radium owned by the laboratory was originally presented to Madam Curie in the form of uranium residues. Only about 1 gram of radium has been extracted in any degree of purity, but a large quantity of residue still remains on hand. This gram of radium is used for purely scientific research and not for medical purposes.

The English Radium Institute was founded by Sir Ernest Cassel and Viscount Iveagh, who gave a large sum for its endowment. The institute does not own a mine nor extract radium from ore, but buys whatever refined radium salts it needs. It has at its disposal probably somewhere between 1 and 2 grams of radium chloride. Its work is entirely confined to the medical and other scientific uses of radium, primarily the former. The annual report, which has recently been issued, has already been mentioned.

MARKET VALUE OF RADIUM SALTS.

The price of radium salts varies to some extent, depending upon the quantity purchased and also from whom purchased. The average price for radium bromide in small quantities is \$70 per milligram. Some makers charge as high as \$100 per milligram. This, however, is for a specially pure and guaranteed product. Lower-grade material has a correspondingly lower price. For example, a sample containing 50 per cent radium bromide is worth half as much per milligram as one which is guaranteed to be perfectly pure. A price of \$70 per milligram corresponds to \$70,000 per gram, or \$2,000,000 per ounce.

There is hardly any material on the open market to-day that offers better chances for fraud in connection with its sale. The reason of this is that the average chemist is unable to tell whether the radium salt is pure, and to what degree it is below the guaranteed standard. In the earlier days of the sale of radium a radioactivity standard was used. Under this standard any material that had a radioactivity of two million^a was represented as pure. Material of lower radioactivity contained a proportionate amount of radium. The standard was absolutely arbitrary and for temporary purposes was sufficiently exact, but conditions have changed. A person purchasing radium should insist on receiving a guarantee, stating exactly what per cent of radium chloride or radium bromide the sample contains. Upon no other basis can a purchaser be sure of obtaining a standard article.

^a The radium was supposed to be 2,000,000 times as radioactive as a similar weight of pure uranium oxide.

Another point which it is necessary to bear in mind is that in radium chloride and radium bromide the percentage of metallic radium is very different, the chloride containing 76 per cent radium and the bromide less than 59 per cent. It can therefore be seen that the values of a milligram each of the two salts, based upon the metallic radium content, are different. The same price per milligram should not be paid for bromide as for chloride. All dealers should sell their products on the basis of the proportion of metallic radium present.

With the large increase in production of radium during the last two years, the question of a market and the continuance of the present prices is a very pertinent subject. Undoubtedly it is now more difficult to dispose of radium than it was a few years ago, due to this increase in production. As already stated, the future market will depend very largely upon the successful use of radium for medical purposes. If it can be demonstrated to the satisfaction of the medical profession as a whole that it has a decided use in any one disease, the demand for radium will not only equal the present supply but will equal any supply that is likely to be put upon the market in the future. For example, if it can be proven that treatment with the radium emanation is the best cure for rheumatism, since 1 gram of radium chloride is enough for treating only 170 persons continuously, it can be seen that no likely supply in the immediate future could equal the demand. Therefore a supply of radium salts sufficient to meet the needs of an adequate investigation of the medicinal properties of radium should either be placed in the care of a radium institute in this country, or made available for use in some of our large hospitals. The radium in the Colorado and Utah ores should not be sent abroad in the future as it has been in the past. This country should receive the scientific and medical benefits which may be expected from its retention and use on this side of the Atlantic.

RADIOACTIVE METHODS FOR TESTING ORES.

The matter of sorting uranium ores correctly is most important to the practical miner. From the descriptions of the ore bodies in Paradox Valley and in Utah, one can readily see that it is sometimes extremely difficult to tell what is and what is not a shipping ore. The fact that in many cases long hauls are necessary to carry the ore to the railroad makes it doubly important that no ore should be shipped that can not be marketed. The miners learn by experience to distinguish high-grade from low-grade ore, but as the appearance of the ore varies in different localities, a sorter who is successful in one place may make serious errors in another. In addition, as already stated, a number of analytical mistakes have been made by some assayers with rather serious consequences to some of the shippers. It there-

fore becomes exceedingly important to have some quick method by which a mine operator can tell whether the uranium oxide content of his ore is or is not above the minimum of 2 per cent. This can be readily accomplished by an intelligent person with a little practice by means of an electroscope.

METHOD OF USING THE ELECTROSCOPE FOR APPROXIMATE DETERMINATIONS.

A suitable electroscope usually consists of two compartments; one above containing a suspended gold leaf in front of which is attached a reading microscope, and one below in which the ore to be tested is placed. Usually the leaf is electrically charged by means of a piece of vulcanite rubbed on the sleeve of the coat, the charge causing the leaf to rise; then the natural leak of electricity from the leaf is noted on the scale, and calculated as a certain number of divisions per minute. The ore is then placed in the compartment below and the leak of the leaf noted as before. If the ore contains uranium and radium, the rate at which the leaf falls will always be faster than the natural leak of the instrument itself. This is due to the fact that the rays given off by the radioactive material ionize the air in the compartment in which it is placed, and if the leaf has been charged positively the negative ions will be attracted to the leaf system and will discharge the charge which has been imparted to it. This ability to discharge electricity is the means by which we recognize radioactive substances.

There are a number of precautions, however, to be taken in making such measurements. First, the illumination during the taking of the readings should be constant. It is therefore much better to make all determinations at night, when an artificial light can be placed at a definite distance behind the electroscope. Second, readings should always be taken between the same points on the scale. Third, in comparing two ores their physical condition should be as nearly as possible the same. This may be roughly done by passing them through the same mesh sieve, preferably 40 or 60 mesh. Of course, every particle of the ore must be ground until it finally passes this sieve, otherwise there will be a partial concentration of the radioactive material in the finer portion. The same weight should be taken and the same surface should be exposed in the electroscope. A balance is inconvenient in a mining camp, but approximately uniform quantities of ore may be obtained in the following way:

In a brass plate about one-fourth inch thick, of a size to fit into the bottom compartment of the electroscope, should be cut by means of a lathe a circular depression one-eighth inch deep and about 2 or 3 inches in diameter. This can be done by any brass worker. The bottom and sides of the depression should be perfectly smooth. The ore

to be tested is poured into the depression, the plate tapped gently so as to settle the ore, and then by passing the edge of a flat piece of metal across the surface of the plate the extra ore is wiped off and the depression left exactly filled with ore with a perfectly flat surface.

In this manner a fairly uniform weight of material is obtained for comparison and the surface exposed in the electroscope is approximately constant. Of course, the density of the ores tested varies, but the method is accurate enough to give approximate results.

The plate with the ore is introduced into the bottom compartment of the electroscope and a reading taken. The ore is removed and replaced by a sample of carnotite of known uranium content which serves as a standard. This sample, of course, is passed through the same mesh sieve as the sample being tested. The relative radioactivities; that is, the rates at which the leaf falls, are roughly proportional to the amount of U_3O_8 present, it being assumed that the ratio of radium to uranium in the two ores is the same. The following example will show how to make a calculation:

Method of calculating percentage of U_3O_8 .

Natural leak of instrument=5 divisions in 10 minutes.

Natural leak of instrument=0.5 division per minute.

Rate of fall of leaf with standard ore (3 per cent U_3O_8)=48.5 divisions per minute.

Rate of fall of leaf with ore to be tested=36.5 divisions per minute.

Subtracting from each of these figures the natural leak, 0.5 divisions per minute, give 48 and 36. The percentage of U_3O_8 in the ore will then be $\frac{36 \times 3}{48} = 2.2$.

If the natural leak is as low as 0.5 division per minute and the radioactivities of the samples are as high as those indicated in the above experiment, the natural leak can be neglected, as the error from it is less than the probable experimental error.

The following actual comparison of the radioactivity of two carnotite ores, the uranium content of which was determined by analysis, will show the probable error in work that is carefully done.

Comparison of two carnotite ores.

	First ore.	Second ore.
Percentage of U_3O_8	2.02	3.16
Divisions per minute.....	7.4	12.0

If the 2.02 per cent ore be taken as the standard of comparison, the electroscopic results indicate that the second ore contains 3.27 per cent U_3O_8 instead of 3.16 per cent, the agreement being close enough for practical purposes.

The method outlined above is only approximately correct, because every sample of radioactive material has a different emanating power; that is, it occludes the emanation to a different degree. Other factors, as already explained, affect the result and are difficult to control.

However, with care and a little practice the electroscopic method can be successfully used to give an approximate indication of the uranium content of the ore. The total time required for a determination, including the grinding of the material, should not be more than one-half hour.

If the radium content of the ore is desired, in order to obtain an accurate determination, it is necessary to use proper, if small laboratory facilities, a different type of electroscope, and a little more skill in manipulation than is necessary for getting a rough approximation of the radioactivity by the method already described. There are a number of electroscopes on the market which can be used. A suitable one is shown in Plate IV, *B*. The upper portion, which contains the leaf system, can be unscrewed and removed from the lower portion, which allows repairs to the leaf system to be readily made. The insulation is either sulphur or amber. As the lower compartment must be absolutely airtight and it is difficult to get a satisfactory contact between the sulphur or amber and the brass, a cement should be used. One made by melting a mixture of rosin and black rubber is satisfactory. The lower compartment contains a thin brass plate or cylinder connected by means of a brass wire with the leaf system in the upper compartment. The charging device is plainly shown in the figure. The lower compartment can be partly exhausted and a radioactive gas run into it.

METHOD OF USING THE ELECTROSCOPE FOR EXACT DETERMINATIONS.

STANDARDIZATION OF THE ELECTROSCOPE.

The electroscope has to be standardized; that is, the effect on it of the emanation that is in equilibrium with a given quantity of radium must be determined. When this is known, the radioactivity of the gas can be calculated by comparison with the effect obtained from the standard radium emanation. The method of standardization is based upon the fact that in any unaltered uranium mineral the ratio of the radium present to uranium is constant. Rutherford and Boltwood^a have determined this ratio and found that 1 gram of uranium is in radioactive equilibrium with 3.4×10^{-7} grams of radium. This means that a quantity of pitchblende, or any other unaltered uranium mineral, which contains 2,000 pounds of metallic uranium, or 2,360 pounds of uranium oxide, will have in it 308 mg. of metallic radium, which is equivalent to 404 mg. of radium chloride.^b If, then, the emanation from a small weighed quantity of a uranium mineral, in which the percentage of uranium is known by chemical

^a Rutherford, E., and Boltwood, B. B., The relative proportion of radium and uranium in radioactive minerals: *Am. Jour. Sci.*, ser. 4, vol. 22, 1906, p. 1.

^b According to these figures 1 ton of 30 per cent ore would contain 102.7 mg. of radium calculated as chloride.

analysis, be introduced into the electroscope and the rate of fall of the leaf noted, a constant for the electroscope may be calculated from the data that shall express the uranium (or radium) required to cause the leaf to fall one division of the scale in a unit of time. The electroscope is standardized as follows:

The emanation is separated and collected with the apparatus^a shown in figure 2. A small quantity of a standard sample of powdered uraninite or pitchblende of known uranium content is weighed out and put in the flask F, which has a capacity of about 50 c. c. Usually a quantity that contains 10 mg. of metallic uranium is convenient. A rubber stopper, fitted with a small dropping funnel and a short delivery tube, is then inserted in the flask and connections are made with the gas burette B, which has been previously filled with freshly boiled hot distilled water containing a little sodium hydroxide. Nitric acid (1:1) is then poured into the dropping funnel, the leveling reservoir of the gas burette is lowered below the level of the acid in the funnel, and then, by opening the pinchcock C and the stopcock of the funnel, most of the acid is allowed to flow into the flask. The stopcock is then closed, the leveling reservoir is replaced, and the flask is gently heated until the pitchblende is dissolved. By withdrawing the flame for a few moments, sufficient water is allowed to flow into the flask to continue the boiling for 10 minutes. The gas collected in the burette is then introduced into the electroscope which has previously been partly exhausted. After three hours, which is the time required for the radioactivity to attain a maximum, the rate of leak is determined.

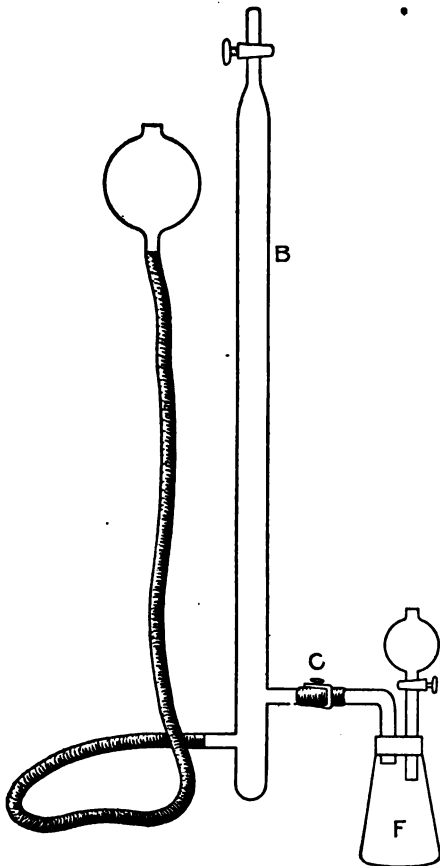


FIGURE 2.—Apparatus for separating emanation from uraninite.

^a Schlundt, Hermann, and Moore, R. B., Radio-activity of the thermal waters of Yellowstone National Park: U. S. Geol. Survey, Bull. 395, p. 19.

CORRECTION FOR STANDARDIZATION.

A small correction, first indicated by Boltwood,^a has to be made for each sample of pitchblende used for standardization, but 2 or 3 grams of the mineral, once this correction has been ascertained, will last for years. The correction is made as follows:

About 0.1 gram of the mineral is weighed and introduced into a small tube closed at each end by means of a piece of rubber tubing and a clip. The tube is allowed to stand until the emanation given out by the pitchblende is in equilibrium. One end of the tube is then connected with the previously exhausted electroscope and a current of air passed gently through the tube into the electroscope. This carries into the instrument the emanation that has been given out naturally in the cold by the pitchblende. The effect on the electroscope is noted and is compared with the effect from the emanation occluded by the pitchblende, the emanation being liberated by boiling with acid as described above. If one-tenth of the emanation is given out in the cold one-tenth more material must be used. For example, instead of taking a quantity of the pitchblende that contains 10 mg. of metallic uranium, a quantity must be used that contains 11 mg., but the calculations are made as though a quantity containing 10 mg. had been used.

METHOD FOR EXACT DETERMINATION OF RADIUM.

After the electroscope has been standardized and it is desired to determine the radium content of a mineral, a definite quantity of the mineral, depending on its uranium content, is fused with four or five times its weight of mixed sodium and potassium carbonates. In the case of a carnotite containing 2 to 4 per cent U_3O_8 , about $\frac{1}{2}$ gram of material should be used. The fused mass is lixiviated with water and washed with dilute sodium carbonate solution. The insoluble residue is dissolved in dilute hydrochloric acid. The alkaline and acid solutions are placed in separate flasks, which are then corked. Through the stopper of each flask passes a glass tube, which is closed by a piece of rubber tubing fitted with a small clip. On standing for one month, the radium emanation is re-formed by the radium in solution and is in equilibrium with the radium; that is, the maximum amount has been re-formed. The emanation in the two flasks is boiled off into the gas burette in a manner similar to that already described, but without the addition of acid, and the air containing the emanation is introduced into the electroscope. After 3 hours a reading is taken and from the data thus obtained the percentage of radium in the sample can be determined. An example of this method follows:

^a Boltwood, B. B., On the radioactivity of natural waters: *Am. Jour. Sci.*, vol. 18, 1904, p. 381.

Sample determination of radium content of an ore.

A sample of pitchblende loses 10 per cent of its emanation at room temperatures. It contains 50 per cent metallic uranium. Therefore 22 mg. of the ore will, on dissolving in acid, liberate emanation in equilibrium with 10 mg. (0.01 gram) of metallic uranium. This emanation, 3 hours after introduction into the electroscope, causes the leaf to fall at the rate of 40.5 divisions per minute. The natural leak (0.5 divisions per minute) subtracted from this leaves 40 divisions per minute due to the emanation. Therefore the fall of one division per minute represents the total emanation associated with $\frac{.01}{40} = 2.5 \times 10^{-4}$ grams of uranium in the mineral. This is the "constant" for the electroscope.

One gram of ore is fused with fusion mixture as already described. At the end of a month the emanation obtained from the two solutions is introduced into the electroscope. After 3 hours the rate of fall of the leaf is 18.5 divisions per minute. Subtracting the natural leak (0.5) leaves 18 divisions per minute. Therefore 1 gram of the ore contains $18 \times 2.5 \times 10^{-4} = 45 \times 10^{-4}$ grams of uranium.^a As one gram of uranium is in radioactive equilibrium with 3.4×10^{-7} grams of radium, 1 gram of the mineral will contain $(45 \times 10^{-4}) \times (3.4 \times 10^{-7}) = 1.53 \times 10^{-9}$ grams of radium.

COMMERCIAL METHODS OF TREATMENT OF ORES.

Almost all the commercial methods which have been proposed for treating uranium ores, or ores carrying both uranium and vanadium, have been for the extraction of the uranium and vanadium only, without reference to the recovery of the radium. Some of these methods have been used in concentration only and not for obtaining refined products. In such cases the radium has usually been lost. The reason that no attempts have been made until recently to extract the radium, in addition to the other two metals, is probably because of operators not knowing the best methods to use, a lack of capital, or to a hesitation in entering a new and little-known field. The ores have been purchased abroad mainly for their radium content and the profit has been largely in connection with the extraction of the radium. This can be readily shown by considering gross returns on the refined products that would be obtained from working up a carnotite ore containing 3 per cent U_3O_8 and 4 per cent V_2O_5 , which are fairly typical values for ore from the Paradox district.

Value of uranium and vanadium per ton of ore.^c

3 per cent U_3O_8 =60 pounds per ton; at \$2.50 per pound	= \$150
4 per cent V_2O_5 =80 pounds per ton; at \$0.75 per pound	= 60
60 pounds of U_3O_8 =8.1 mg. ^b of radium chloride; at \$90 per mg.=	729
Total,	939

^a This is true only when the uranium and radium are in equilibrium. In other cases it represents the theoretical amount of uranium in equilibrium with the radium actually present. In pitchblende, since it is a primary mineral, the ratio of uranium to radium is constant, 1 gram uranium= 3.4×10^{-7} grams of radium, or 2,000 pounds U_3O_8 =342 mg. $RaCl_2$, and if the percentage of uranium is known by analysis, the amount of radium present can be calculated directly. In carnotite and other recent uranium minerals, the equilibrium ratio is not constant and the radium present has to be found by experiment.

^b The theoretical content of radium chloride from 60 pounds of uranium oxide would be 9.2 mg.; 10 per cent has been allowed for the radium and uranium not being in equilibrium.

^c The figures given do not take into consideration losses in treatment.

The refined uranium and vanadium products from a ton of the above ore would be worth \$210, and the radium chloride extracted from a ton of the same ore would be worth \$729, or more than three times the value of the uranium and vanadium. Therefore a method of extraction that would not be profitable if the uranium and vanadium alone were recovered might yield a profit if it recovered the radium also. It is difficult to give the exact cost of extracting and refining radium salts. This varies in different places, being governed by the cost of labor, chemicals, freight, and other items, but \$20 per milligram is probably a maximum cost in Europe for extracting radium from carnotite ores. Assuming that the market value of radium remains at its present figure, namely, \$90 per milligram for radium chloride, this leaves a wide margin of profit over the necessary cost of production in this country even though the cost be somewhat more than abroad.

The carnotite produced in the United States during the last two years has been the means of a sudden and large increase in the production of radium salts, and the market for radium at present is somewhat uncertain, but if prices should fall the margin is considerable. Any method, therefore, used in this country for the extraction of uranium and vanadium from carnotite ores, or of uranium from pitchblende, should include the saving of the radium in marketable form, even though it does not involve final purification of radium salts. The purification, however, of radium salts of medium activity, such as those containing from 10 to 50 per cent radium chloride, is not difficult. It is difficult to obtain chemically pure radium salts, but these in most cases are not at all necessary, the lower-grade material being just as valuable, if not more so, for most medical and other scientific purposes.

BLEEKER PROCESS FOR THE RECOVERY OF VANADIUM.

The method of treatment suggested by W. F. Bleeker^a involves the production of copper, iron, or lead vanadate from vanadiferous ores and has for its object the expeditious and clean separation of the vanadium constituents of the ores. The process includes two general stages—the production of an approximately neutral vanadium solution, and the precipitation of this solution with the metallic substance. The first stage is performed in five successive steps: The ore is pulverized to a suitable degree of fineness; the pulverous matter is roasted after having been mixed with a flux to render the vanadium constituents soluble after roasting; the roasted product is leached with water to dissolve the alkali vanadate or vanadyl salts; the residue is leached with a dilute acid solution to dissolve any

^a United States patent 1,015,489.

remaining vanadium compounds; and the two solutions are mixed to obtain a neutral vanadium solution. The flux by which the vanadium constituents of the ore are rendered soluble after roasting is composed of sodium chloride and a fixed alkali, such as potassium hydroxide, sodium hydroxide, potassium carbonate, or sodium carbonate, preference being given to the potassium hydroxide. The product, after being roasted with this flux, is leached with water to dissolve the soluble alkali vanadates, and this rich alkaline liquor is led to storage tanks. The residue is again leached, but with a dilute mineral acid, such as hydrochloric, for the purpose of dissolving the vanadates insoluble in water, leaving a residue practically free from vanadium. The alkaline and acid liquors are mixed in such proportions as will produce a neutral solution. Copper sulphate or any other copper salt is added to the solution to make an insoluble precipitate of copper vanadate which is easily filtered.

This process was devised for the treatment of roscoelite ores for the Rare Metals Mining & Milling Co. These ores do not carry uranium. The process might, however, be used with carnotite ores and by a slight modification of it a large part of the radium recovered. A test of a small sample of carnotite ore made by the Bureau of Mines, in which two parts of ore were sintered with one part of sodium carbonate and one part of sodium chloride, showed that the total radium content present in the alkaline solution, the acid solution, and the insoluble residue was as follows:

Distribution of radium content of ore in Bleeker process.

	Per cent.
Alkaline solution.....	1.1
Acid solution.....	72.0
Residue.....	26.9
	100.0

The radium could be readily obtained from the acid solution by precipitating it with a moderate quantity of barium sulphate. This will precipitate the radium as radium sulphate, from which the radium can be recovered either by fusion with fusion mixture or by leaching with sodium carbonate, as described under methods of obtaining radium from pitchblende residues (p. 79). The loss indicated by the figures above is altogether too large for a commercial plant, but the yield might possibly be improved by using different proportions and by changing the conditions.

HAYNES-ENGLE PROCESS FOR THE RECOVERY OF URANIUM AND VANADIUM.

The process ^a of J. H. Haynes and W. D. Engle involves the treatment of ore containing either uranium or vanadium, or both of these metals. The ore is first crushed to twelve mesh, and is then boiled with a solution of alkaline carbonate, preferably sodium carbonate or potassium carbonate, until the uranium or vanadium, or both, in the ore is dissolved. The strength of the sodium-carbonate solution and the length of time necessary to boil are determined by the proportion of uranium and vanadium in the ore and will probably vary considerably. The originators of the process claim, however, that 100 pounds of sodium carbonate per ton of ore for each 1 per cent of uranium and vanadium, or either, present will give good results. Ordinarily the time required for boiling should be about one hour. After the uranium and vanadium, or either, are dissolved, the clear solution is drained into a separate tank. The uranium is precipitated as sodium uranate by the addition of sodium hydroxide to the solution. This precipitate is removed from the solution, which contains all of the vanadium. From this solution, either with or without neutralization, the vanadium is precipitated as calcium vanadate by the addition of water-slaked lime.

When the process was in actual operation with carnotite ores an extraction of 80 per cent of the uranium and 60 to 65 per cent of the vanadium was obtained.

The main object of this method is the chemical concentration of carnotite ores, involving the recovery of the uranium and vanadium. The radium, all of which passes into the insoluble residue, could be removed by the following treatment: After the residue from the alkaline solution has been thoroughly washed in order to remove all sulphates, that portion of the radium that has been converted into radium carbonate, which is insoluble in the alkaline solution, could be leached from the residue by dilute hydrochloric acid, and the radium chloride could be recovered from this solution without difficulty. If any of the radium still remained in the residue, it could be recovered by boiling a second time with the carbonate solution, washing as before, and leaching again with dilute acid. The commercial success of such a method would depend upon whether or not all of the radium could be leached out by the first treatment with acid, the proportion of sulphates and alkaline earths in the ore largely determining this point.^b

^a United States patent 803,839.

^b A patent has been recently taken out by W. F. Bleeker (U. S. patent 1,065,581), involving the recovery of the radium as indicated.

KOENIG PROCESS FOR THE RECOVERY OF VANADIUM.

G. A. Koenig's process^a relates to improvements and processes for the recovery of vanadium from its ores particularly the vanadiferous sandstone of southwestern Colorado and other places. The method is based upon the fact that roscoelite, or ores of vanadium containing the same mineral substances, can be completely decomposed and dissolved by the action of a dilute solution of sulphuric, hydrochloric, or other acid under proper heat and pressure.

In practice, the originator uses a solution containing about 20 per cent sulphuric, hydrochloric, or other acid, at a temperature of about 200° C. and under a pressure of about 225 pounds per square inch, and claims that this process will completely decompose and dissolve the roscoelite within a few hours. The filtered solution is evaporated to a mushy consistence, placed in a retort or muffle and heated gradually to a bright red heat. This heating drives out the acid, which may be recovered in any suitable manner. The residue is a mixture of the oxides of vanadium, aluminum, manganese, iron, and other metals that may be present in the ore. When sulphuric acid is used sulphates of calcium, potassium, and other sulphates are also present. The mixture of oxides and sulphates is mixed with the proper quantity of sodium carbonate and is roasted at red heat in an oxidizing flame either with or without the addition of oxidizing agents. The roasted mass is disintegrated with boiling water and, while still in the boiler, is treated with carbon dioxide to precipitate the aluminum as aluminum hydroxide.

The acid methods and their value in the extraction of radium will be discussed later.

FISCHER PROCESS OF EXTRACTING VANADIUM FROM CARNOTITE.

In Siegfried Fischer's method of extracting vanadium from carnotite the ore is boiled with a solution of sodium or potassium hydroxide. The process comprises three distinct steps: The conversion of vanadium compounds insoluble in water into soluble form, leaching and separating vanadium from uranium, and precipitating and drying the vanadium compound. It is claimed that the dried product is ready for reduction to ferrovanadium and that the uranium residue is in marketable condition. By this process, working on crude ores, Fischer claims an extraction of 65 to 67 per cent of the vanadium present, and from concentrates carrying from 9 to 16 per cent vanadium an extraction of 93 to 94.6 per cent.

In this method the uranium would remain in the insoluble residue as sodium uranate with the radium, and the value of the sodium-

^a United States patent 986,180.

hydroxide leach would be simply to extract the vanadium. An appreciable proportion of the sodium hydroxide would react with the silica present, involving a large waste of material. It does not seem to be a desirable method for the treatment of carnotite ores when the extraction of the radium is desired.

METHOD USED BY PRIMOS CHEMICAL CO.

The Primos Chemical Co., of Newmire, Colo., and Primos, Pa., roast roscoelite ore with common salt. The ore contains almost no uranium and no attempt is made to recover the minute quantity found in some of it. The ore is weighed and mixed with salt. The mixture is then coarsely ground and run into a drier which reduces the moisture content to 1 per cent. The material, which comes out somewhat caked, is ground to 20 mesh and roasted in a furnace for about three hours. The roasted material is delivered at the bottom of the furnace to conveyors, which carry it to lixiviation tanks, where it is treated with water and filtered. The vanadium is now in solution as sodium vanadate. A solution of ferrous sulphate is added to the filtrate, and the vanadium is precipitated as vanadate of iron. The precipitate is filtered out of the solution and dried. The dried material is sent to Pennsylvania, where it is reduced to ferrovandium, probably by the Goldschmidt process. A complete extraction of vanadium is not obtained by this method.

If carnotite were treated in a similar manner the solution of the vanadium as sodium vanadate might be accomplished in the same way. The uranium might be converted into sodium uranate, which would remain in the residue. The radium would also be in the residue.

For this reason the method does not appear to be applicable to carnotite ores.

FLECK METHOD OF EXTRACTING URANIUM AND VANADIUM.

The American Rare Metals Co., of Denver, Colo., uses a method of extraction originated by Herman Fleck. The company's plant is in the McIntyre district south of Paradox Valley, and the crude sulphuric acid required in the process is hauled in wagons from Dolores. The finely crushed ore is treated with dilute sulphuric acid, which dissolves the uranium, vanadium, copper, and iron contained in the ore. The solution is decanted from the slime and sulphur dioxide gas is passed through it, reducing the iron and vanadium compounds to ferrous and vanadous forms. A calculated quantity of pulverized limestone (the rock is obtained a short distance from the plant) is then added until the metals begin to separate, calcium sulphate being precipitated. The solution is then decanted or filtered from this sulphate and the precipitation of the metals is completed by boiling with more pulverized limestone. A concentrate is thus

obtained carrying about 20 per cent of uranium and vanadium calculated as oxides. Improvements in the process are contemplated by which a still higher concentration can be obtained.

The radium, of course, is in the undissolved slimes and the radium concentrate is obtained by fractionation of these slimes. A product carrying as high as 100 milligrams of radium per ton has been produced.

This process is in operation at the present time and recently a shipment of uranium, vanadium, and radium concentrates was made from Dolores.

RADCLIFF METHOD FOR COMPLEX RADIUM ORES.

Sidney Radcliff's method^a relates to the economic commercial treatment of complex radium ores for the separate recovery as marketable products of the following substances: (a) Radium, as radium and barium sulphates; (b) uranium, as oxide or uranate; (c) the "acid earths," such as tantalum, niobium, titanium, as oxides; (d) the "rare earths," such as cerium, thorium, lanthanum, and didymium, as oxides.

The crushed ore or concentrates is fused with acid sodium sulphate. A powerful decomposing and oxidizing reagent is added to the fusion. The fused mass is cooled, pulverized, and thoroughly lixiviated and agitated with water, which dissolves certain constituents. The solution, which contains sulphates of radium, barium, and other elements in suspension, is led to settling tanks. The suspended and dissolved matter is treated as hereafter described for the recovery of the valuable constituents. In this manner the quantity of material to be precipitated with carbonate of soda is greatly reduced, thereby lessening the cost of treatment.

The various steps of the treatment are as follows:

The ore or the concentrate is crushed to pass a 30 to 40 mesh sieve.

The crushed ore is fused in a reverberatory or other suitable furnace with about $2\frac{1}{2}$ times its weight of acid sodium sulphate. After the charge has been fused, and while the mass is still fluid, sodium chloride (10 or 15 per cent of the weight of the ore) is added and well rabbled. The addition of the sodium chloride and its reaction with or in the presence of the fused acid sodium sulphate causes a powerful decomposing and oxidizing effect and changes any ferrous sulphate to ferric sulphate.

The fused product is tapped from the furnace in the liquid state, cooled, crushed to powder, charged into suitable vats containing warm water, and agitated for some time. Most of the uranium, iron, and "rare earths," together with part of the titanium, niobium, and tantalum, go into solution. The radium is in the form of a sulphate

^a United States patent 1,049,145.

which is insoluble and remains in suspension along with the sulphates of lime, lead, and barium, and the fine particles of gangue material.

The turbid liquid is rapidly siphoned into suitable settling vats. The coarse residues which remain in the bottom of the vat are washed several times with warm water and then rejected, the washings being passed into the settling vats.

From the settling vats two products—(a) solution and (b) fine slime—are obtained.

The solution (a) contains in the case of certain ores iron, aluminum, chromium, and uranium compounds, as well as compounds of the "acid earths" and "rare earths." Sodium carbonate is added, but not in excess, to the solution and all the above-mentioned elements are precipitated. The precipitate is removed by means of a vacuum filter and the clear liquid rejected. The precipitate recovered on the filter is boiled with an excess of a solution of sodium carbonate, which causes the uranium to pass into solution. The solution is filtered off and the uranium is then recovered as sodium uranate by the addition of sulphuric acid or caustic soda. The balance of the precipitate, which still contains, in addition to compounds of iron and aluminum, "acid earths" and "rare earths," is treated with moderately dilute sulphuric acid, which dissolves all of the precipitate with the exception of the "acid earths." The latter are filtered off, washed, and ignited. The washings are added to the sulphuric acid filtrate. This filtrate is treated with oxalic acid, which precipitates the "rare earths" as oxalates. These oxalates are then washed, dried, and ignited.

The fine slimes (b) contain most of the radium and are treated according to the usual methods. The crude radium and barium sulphates are obtained in the ordinary way.

The objections to this method, in the treatment of low-grade carnotite ores, are the inconvenience and cost of fusing an ore that contains as much as 95 per cent of silica, iron, and calcium compounds in order to obtain the average 5 per cent of uranium and vanadium compounds present.

TREATMENT OF CARNOTITE ORES WITH NITRIC OR HYDROCHLORIC ACID.

The simplest way to extract uranium, vanadium, and radium from a carnotite ore is to treat the ore or concentrate directly with boiling concentrated nitric or muriatic acid. Even the vanadium and silica combinations can be decomposed by boiling with these acids (1:1) for an hour, nitric acid giving the better results. Carnotite itself is soluble in cold dilute hydrochloric or nitric acid. The practicability of such a process depends on its cost and the possibility of readily recovering the radium. The first step is what counts in the main in treating these ores. After that, the separation of the uranium

and vanadium is a simple matter, several methods being available. Nitric acid would be absolutely out of the question for use in a commercial process if the vanadium and uranium alone were recovered, but the large additional cost of nitric acid over sulphuric and hydrochloric acids might be more than offset by the saving of labor and time if by its use a convenient and easy method of extracting the radium in addition to the uranium and vanadium could be devised. The price of hydrochloric acid being very little more than that of sulphuric, a small additional saving in the manufacturing process will justify its use instead of the latter acid.

Some tests made on a small scale on a typical sample of carnotite ore from Paradox Valley, containing 2 per cent U_3O_8 and about $2\frac{1}{2}$ per cent V_2O_5 , showed the following extraction of radium.

Extraction of radium from a carnotite ore.

With hot concentrated commercial muriatic acid, 97.1 per cent of the radium in the ore went into solution and 2.9 per cent remained in the residue.

With hot dilute (1 : 2) muriatic acid 88.4 per cent was dissolved in the acid and 11.6 per cent remained in the residue.

With hot, concentrated, commercial nitric acid, 97.5 per cent of the radium went into solution and 2.5 per cent remained in the residue.

With hot dilute (1 : 2) nitric acid, 96.8 per cent of the radium was dissolved and 3.2 per cent was left in the residue.

The concentrated acids in both cases gave the better extraction, although the dilute nitric acid was not much inferior to the concentrated. The radium can be easily recovered by further dilution and by the precipitation of barium sulphate in the solution. The radium which is carried down with the barium sulphate as radium sulphate can be recovered by leaching the precipitate with a solution of sodium carbonate, or by reducing the mixed sulphates to sulphides by heating the precipitate in a furnace in a current of coal gas, then dissolving the sulphides in hydrochloric acid and fractionating. Although a better extraction is obtained with nitric acid, its use in addition to the increased cost has some disadvantages. Nitric acid is unpleasant and difficult to handle, although porcelain-lined ware is now made which withstands the action of hot nitric acid exceedingly well. Also a precipitate of barium sulphate does not carry down the radium as well from dilute nitric acid solutions as from dilute hydrochloric acid solutions. The figures given above were obtained with the same sample of ore, which contained only a small proportion of sulphate, although the acids themselves contained some sulphate. In the case of an ore with a large gypsum content, the extraction, especially with hydrochloric acid, would be much lower than it was in this case.

The same acid could be used in treating successive portions of ore until the strength became reduced to a certain point. In addition

to their uranium and vanadium content, all of these ores carry a certain proportion of iron and calcium, as well as traces of other metals. The total acid-soluble material in the ore treated as mentioned above was about 12 per cent. For some ores this figure would be higher and for others lower. The results of further investigations in this direction are to be published by the Bureau of Mines.

Hot concentrated nitric acid can be used for dissolving pitchblende and changing both the uranium and radium into soluble form, provided the ore does not carry too much iron or copper pyrite. After separation and decantation from the insoluble residue, the acid can be diluted and the radium recovered with barium sulphate by precipitation. If this method is not desirable, the radium can be obtained mixed with salts of barium, strontium, or calcium, according to the method of separation that is chosen.

BLEEKER PROCESS OF SEPARATING URANIUM FROM VANADIUM.

A process ^a is described by W. F. Bleeker for the separation of vanadium from uranium. It is stated that this process is applicable for treating any solution of uranium and vanadium containing an alkaline carbonate, such as sodium carbonate, ammonium carbonate, or potassium carbonate. The solution is first heated to a temperature preferably not exceeding 90° C., and heated sodium hydroxide is added in sufficient quantity to precipitate uranium as a mixture of sodium uranate and uranyl hydrate. This precipitate carries vanadium and the resultant solution also carries some uranium which may be recovered by any suitable subsequent treatment. The precipitate is filtered and washed with water, dissolved in acid, preferably sulphuric, forming a solution of uranyl sulphate. This solution is treated with an excess of sodium carbonate so as to make it slightly alkaline. It is then electrolyzed in a tank, the anodes being of any desired active metal, such as iron, copper, or nickel, and the vanadium is precipitated as the vanadate of the anode metal. For example, if a nickel anode is used, the product would be vanadate of nickel. The uranium remains in solution. After the solution has been electrolyzed long enough to precipitate all of the vanadium, it is filtered, and the uranium salts, free from vanadium, may be precipitated by any well-known process.

PROCESS OF EXTRACTING VANADIC ACID FROM COPPER VANADATE.

Bleeker also describes a process ^b of extracting vanadic acid from insoluble copper vanadate. Copper vanadate is decomposed by a dilute mineral acid, preferably sulphuric, the product obtained being an acid solution of copper and vanadium. The insoluble vanadic

^a United States patent 1,050,796.

^b United States patent 1,049,330.

acid (V_2O_5), which is about 80 to 90 per cent pure, is separated by filtration, leaving copper and vanadic acid in solution ($Cu + V_2O_5 + H_2SO_4$). The copper is extracted from this solution, and the vanadic acid obtained by electrolysis. The cathode is of copper, and the anode is preferably carbon or platinum. The copper is deposited on the cathode, and the vanadium remains in solution ($V_2O_5 + H_2SO_4$). By evaporating all or part of the sulphuric acid from this solution, the vanadic acid may be recovered. The mother liquor upon dilution is still strong enough to be used in the process of producing copper vanadate.

AUSTRIAN METHOD OF EXTRACTING RADIUM FROM PITCH-BLENDE.

In the method used for treating pitchblende in Austria, the pitchblende is fused with sodium sulphate, and the uranium is thus changed to sodium uranate, which can be dissolved by means of dilute sulphuric acid. The residue contains all of the radium. Before the discovery of radium this residue was considered to be a waste product and was thrown away. The extraction of the radium from the accumulated residue is well described by Haitinger and Ulrich ^a and is, with some minor changes, similar to the method now used by the Austrian Government. The work was done in the laboratory of the Austrian Incandescent Gaslight and Electric Co., and 10,000 kilograms (22,000 pounds) of pitchblende residues were treated, which represented about 30,000 kilograms (66,000 pounds) of pitchblende containing 53.4 per cent of U_3O_8 . Moisture in the various shipments varied from 10.3 to 18.4 per cent. This moisture was driven off at a temperature of $105^\circ C.$, the 10,000 kilograms of residue losing 1,340 kilograms in this way. The work took two years, due to time spent on analyses at the start and to experiments to obtain the best results. The method given was developed as being most suitable to the equipment available. Five thousand kilograms annually could be treated.

The chemical operation involved, first, the decomposition of the residues; second, the removal of the radium sulphate by precipitation; and, third, the solution and concentration of the latter. The first step consisted of digesting the residue with sodium hydroxide, 100 kilograms being boiled for one day with a solution of 50 kilograms of hydroxide in 200 liters of water. Forty per cent of the alkali was converted to sulphate and to silicate. The solution contained some radium, but the total radium so dissolved from the entire 10,000 kilograms of residue represented the radium equivalent of only 10 kilograms. Therefore this solution was thrown away.

^a Haitinger, Ludwig, and Ulrich, Karl, Bericht über die Bearbeitung der Pechblende Rückstände: K. K. Akad. Wissenschaft., 1908, vol. 117, p. 619.

After the boiling the residue was allowed to settle and the liquid was decanted. The residue was washed to remove the greater part of the sulphates, filtering and washing being done in a funnel with a capacity of 100 kilograms, which had a lead suction tube 3 meters long. The vessel containing the washed residue was placed on a water bath and the residue was treated with an equal weight of dilute (1:1) crude hydrochloric acid. After prolonged heating the acid solution was decanted and the residue washed with water. This water was then used to dilute the next portion of acid for a new sample.

Crystals of calcium sulphate and lead chloride formed in the acid solution as it cooled. Neither the solution nor the crystals contained an appreciable quantity of radium, but nearly all the polonium and actinium was in the solution. It was therefore treated with ammonium hydroxide to precipitate the polonium and actinium. The filtrate was not radioactive and was discarded.

The residue from the treatment with crude hydrochloric acid was boiled in a solution of sodium carbonate, the carbonate (made by the ammonia process) being free from sulphates. Fifty kilograms of sodium carbonate in 200 liters of water were used for a 100 kilogram sample. By this treatment a large part of the radium sulphate was converted to radium carbonate. Therefore, in subsequent treatments the solutions had to be kept free from sulphate. The residue was washed till free from all trace of sulphate and then treated with pure hydrochloric acid. The boiling with soda and the treatment with acid was repeated three times. After the third treatment only 2 per cent of the original radium content remained in the residue which was thrown away. The soda extracts were practically free from radium. The washing of each of the various residues consumed from 4 to 6 weeks. The hydrochloric acid extracts, containing nearly all the radium, were united and the radium was precipitated as a sulphate with sulphuric acid. Besides the radium the precipitate contained, of course, the alkali earths, including calcium, much lead containing radio-lead, and a small quantity of rare earths containing actinium. The sulphate precipitate, called crude sulphate, represented from 0.5 to 2 per cent of the weight of the original residue taken.

The crude sulphate was reconverted to carbonate by repeated boiling with sodium carbonate solution. All of the sulphate from any one sample could not be converted and the final residue from the treatments was, therefore, not thrown away, but added to a new portion of crude sulphate. After each carbonate treatment, an extraction with hydrochloric acid was made. The lead chloride formed in the solutions was removed and freed from radium by repeated crystallization in hot water. Sixty kilograms of lead

chloride were thus obtained from the entire 10,000 kilograms of residue. It was saved and treated for its content of radio-lead.

The hydrochloric acid solutions from the crude sulphate were freed completely from lead by hydrogen sulphide and were then evaporated to dryness on a steam bath. The calcium chloride in the residue so obtained was dissolved in concentrated hydrochloric acid, in which barium chloride is only slightly soluble and radium chloride is still less soluble. The residue remaining, called crude chloride, consisted of radium and barium chlorides, with some strontium and calcium chlorides and traces of other impurities.

From this point on the concentration was continued by fractional crystallizations from water solutions. Radium chloride, which is the least soluble of the chlorides, accumulated in the crystals, the foreign matter remaining more and more in the mother liquor. The first fraction was, of course, the richest in radium. Two steps had to be watched in this process—first, the separation from the system of as large a quantity of radium-free barium chloride as was possible; second, the making of a relatively large first fraction. This second step can be taken by temporarily stopping the crystallization of the first series until the crystals of the second series are of sufficient radioactivity to be united with the first.

The crystallizations were all carried out on a steam bath in order to avoid contamination with sulphate, as might have been the case if heating had been done with a direct flame.

Finally, two portions of crystals were obtained, one of about 2 kilograms, containing nearly all of the radium, and the other of about 11 kilograms, with very little radium.

The 2-kilogram portion was treated as raw material for the production of radium chloride free from barium. After this had been crystallized about thirty times, the first fraction of about 9 grams was further crystallized, and the lower fractions were combined into three groups according to their activity. The 9-gram portion was first purified with hydrogen sulphide, removing traces of lead which probably came from the glassware. Further work was conducted in quartz vessels. The salt was dissolved in dilute hydrochloric acid, warmed and allowed to crystallize. Four fractions were so obtained.

Atomic weight determinations were made with three of the fractions, the values obtained being 143.2, 185.2, and 225. The latter represented practically pure radium chloride.

Some of the lower fractions of barium chloride that were poor in radium were converted to the bromide and then fractionated. Only one portion, that which should contain the most radium—that is, the last of the four analogous fractions—was saved. The other portions were reconverted to chloride and added to the main chloride

crystallization system. In all, 3 grams of pure dry radium chloride and 0.236 gram of radium bromide were obtained from 10,000 kilograms of original residue.

ANALYTICAL METHODS FOR URANIUM AND VANADIUM.

Operators in the Colorado and Utah uranium and vanadium fields had considerable difficulty during the past year in having correct analyses made. As a rule, chemists have been able to check their figures on vanadium, but uranium assays from different men have frequently shown widely divergent results. A variation of over 100 per cent has not been uncommon in some cases. As it has been extremely difficult, if not impossible, to sell ore containing less than 2 per cent U_3O_8 , a small error on an ore of this character may mean the difference between selling and not selling. Since the results obtained have been so variable, the foreign buyers are suspicious of such results unless the analyses were made by two or three firms in whom they have confidence. The difficulty has been a tendency to use methods that may work fairly well with some uranium ores, but are of too general a character to be adapted to the large variety of ores associated with carnotite.

METHODS FOR THE ANALYSIS OF CARNOTITE.

The Bureau of Mines has not as yet tested out the analytical methods which are at present in use. Dr. Hillebrand of the Bureau of Standards, whom the writers asked to suggest the best methods of analyzing carnotite, has kindly made the following suggestions:

METHOD FOR THE DETERMINATION OF URANIUM AND VANADIUM.

In the first place, a knowledge of the mineral composition of these ores and of the behavior of their valuable mineral components toward solvents is important. There are several vanadium ores in Colorado and Utah besides pitchblende and the carnotite ores. For instance, there are yellow and green vanadates of copper, barium, and calcium; red vanadates of calcium, sometimes taken for the vanadic acid; and a black ore, rich in vanadium, which carries that element in three states of oxidation. Of all these I have samples and hope to describe them all. The carnotite minerals themselves are perhaps in places accompanied by one or other of the above. Always associated with carnotite, so far as my experience goes, are potassium-vanadium-aluminum silicates, sometimes the green roscelite and again a gray amorphous powder mentioned by me in my paper on carnotite.^a Both of these hold vanadium as V_2O_5 . These silicates are but slowly decomposed by mineral acids, whereas the carnotites dissolve with great ease in even very dilute acids.

This difference of behavior makes it possible to simplify the analysis of such ores, so far at least as the uranium is concerned, for a few minutes' treatment with cold and dilute acid (best nitric) extracts all the uranium almost free from iron and aluminum and accompanied by that vanadium only with which it was combined in carnotite.

^a Hillebrand, W. F., and Ransome, F. L., On carnotite and associated vanadiferous minerals in western Colorado: U. S. Geol. Survey Bull. 262, 1905, pp. 30, 31; Am. Jour. Sci., 4th series, vol. 10, 1900, p. 144.

The remainder of the vanadium in silicate combination can be extracted by boiling for half an hour or so with nitric acid (say, 1:1). The silica of the silicates separates for the most part instead of dissolving in the acid, and in a form that is not gelatinous. It is thus easy to get both the valuable constituents of the ore into solution free from the great mass of quartz and silicate silica without resort to an alkali fusion. In this the vanadium can be determined in any convenient way.

If one prefers, the ore may be at once attacked by the hot acid.

If the cold acid treatment is used my practice is as follows, wherein it must be understood that I aim at the composition of the carnotite and not at the total vanadium in the ore: Evaporate the entire solution to dryness, take up with cold water and filter through a small filter, wash with a little cold water but without transferring the residue to the filter. The filtrate holds almost all the uranium and all the strong bases as nitrates, the vanadium remaining undissolved. The evaporation to dryness with nitric acid may be repeated on the filtrate. If the dry deposit shows a red color, repeat the extraction with water.

Dissolve the residues in a very little nitric acid, transfer the solution by degrees to a capacious porcelain boat, in which it is carried to dryness on the hot plate. Put the boat in a glass tube provided with two large U tubes with water enough in them to seal the bend. Pass HCl gas (from HCl solution and H_2SO_4) through the tube. Instantly brown red fumes of an oxychloride of vanadium come off in great volume. These are held quantitatively by the U tubes whose liquid contents by and by become saturated with HCl.

When the escape of brown fumes ceases, draw the boat out, add nitric acid, evaporate again to dryness, and treat anew with HCl gas, using U tubes with fresh water. Repeat these operations if need be till there is no further evidence of vanadium in the boat. It may be advisable to heat the tube somewhat toward the end of the second or third distillation.

Evaporate the combined distillates with H_2SO_4 to expel the HCl, dilute, introduce H_2S into the blue liquid to complete the reduction of the V_2O_5 and to precipitate Mo and As if present, filter, expel the H_2S , titrate with KMnO_4 , reduce with SO_2 gas (not solution), boil, and titrate again as a check. The SO_2 value is usually a little lower and is the one to be accepted.

Convert the contents of the boat to nitrates and add the solution to the main one. Add H_2S water drop by drop to throw out Cu (or Pb), then a drop or two of H_2SO_4 to precipitate barium; after a time, filter. Uranium can then be determined gravimetrically after separating from alkalies, lime, iron, and aluminum, which separations offer no difficulties if properly carried out. The little phosphorus present is with the ignited U_2O_5 (partly at least) and can be determined and deducted.

These last operations can be shortened, if no arsenic is present, by separating iron and aluminum at once by ammonium carbonate and ammonium sulphide, evaporating the filtrate first with HNO_3 , then with H_2SO_4 , to expel HNO_3 , reducing the sulphate solution with zinc and titrating with KMnO_4 .

All this seems perhaps more complicated than it is in reality, and the operations are not very long, even when repeated. It should be borne in mind, too, that with such valuable ores as these it pays to expend some extra time to insure good results. Moreover, no solid reagents are introduced at any stage of the analysis.

There are some things to be learned, of course, about the distilling operation. The presence of much iron or aluminum retards very much the expulsion of the vanadium, so the method is not to be recommended when these are present in some amount. Care must be taken that there is not enough liquid in the U tube nearest the boat to allow of its being explosively sucked back if the gas current slackens too much before the water becomes saturated. A rapid gas current is desirable. The red fumes liquefy in part in the tube, but if not finally driven over can be washed out with water after withdrawing the boat.

This method will give exact results if properly carried out, but it is probably too much to expect of the average assayer who has to do things in a hurry.

Hillebrand also suggests the following methods of analysis, which are applicable to the carnotite ores, but not to pitchblende. He states that he has not tested them in all their details, and the methods may be varied somewhat to suit the needs of a particular ore when the analyst has intelligent knowledge of the mineral composition and has sound judgment.

METHOD FOR THE DETERMINATION OF VANADIUM.

Roast the ore gently if it carries organic matter, boil for 15 minutes with HCl, filter; ignite insoluble residue, add a few drops H_2SO_4 and evaporate several times with HF. Heat till fumes of H_2SO_4 come off, add more H_2SO_4 , heat, and bring all into solution with H_2O , or fuse the residue with Na_2CO_3 and leach with H_2O and add filtrate to the first one. Evaporate with H_2SO_4 till fumes escape, take up with H_2O and precipitate with H_2S (hot at first, cold later); filter and wash; expel H_2S , oxidize with H_2O_2 , evaporate to fumes of H_2SO_4 , then several times with strong HCl to reduce V_2O_5 to V_2O_4 , take up with H_2O , and titrate with KMnO_4 at 60° to 70° C.

The original directions of Campagne (as applied to steels) are to reduce with HCl before fuming with H_2SO_4 ; otherwise reduction of the V_2O_5 is not quite complete, but experiments by other chemists in this country seem to show that in presence of iron the reduction of V_2O_5 to V_2O_4 in these ores is complete when the order is reversed. Care must be taken, however, in the final fuming with H_2SO_4 not to prolong this beyond the time required to expel the HCl, for there is a slow oxidation of the V_2O_4 after that point is reached.^a

METHOD FOR THE DETERMINATION OF URANIUM.

Treat the ore with cold HCl^b (about 1:1) for 15 minutes, filter, precipitate with H_2S (in heat, and for a long time if arsenic is present), filter, expel H_2S , evaporate to dryness and convert to nitrates by evaporating to dryness with HNO_3 . Treat the residue with cold water and filter at once through a small filter, washing with a little water. This takes most of the uranium nitrate into solution, but little or no vanadium.

Redissolve the residue in nitric acid and evaporate to dryness in a small porcelain dish. Pour NH_4OH over the residue and let stand some time to dissolve most of the V_2O_5 ; filter through a small filter. Redissolve the residue in HNO_3 , evaporate again to dryness, and extract with NH_4OH a second time. Dissolve the final residue in HNO_3 and add to the solution of uranium nitrate. Neutralize this with NH_4OH and add $(\text{NH}_4)_2\text{CO}_3$ and NH_4HS^c . Allow to stand 24 hours in a small stoppered flask, filter, wash with water containing a little $(\text{NH}_4)_2\text{CO}_3$ and NH_4HS . Evaporate the filtrate in a large porcelain or platinum dish, ignite gently, redissolve in HNO_3 , filter if necessary, and then precipitate the uranium with ammonia that is free from carbonate. Filter and wash two or three times with water containing a little carbonate-free ammonia. Redissolve, reprecipitate, and wash as before. Ignite and weigh as

^a If it is known from the character of the ore that the treatment with HCl extracts all the vanadium, the insoluble matter needs no further treatment.

^b The recommendation of HCl, instead of HNO_3 , is based on the possibility of arsenic being present, and of the need of precipitating it from a hot solution by H_2S .

^c The ammonia extraction of the early nitrate residues might perhaps be omitted, but I recommend it strongly because we thereby reduce to a very small amount the V_2O_5 that is finally weighed with the U_3O_8 , and also avoid other possible interferences. The ammoniacal extracts of the V_2O_5 contain, according to my own tests, mere traces of uranium.

crude U_3O_8 . Warm with very little HNO_3 till the U_3O_8 is decomposed, filter from the insoluble residue (it may contain a little silica, iron oxide, and alumina), and deduct the weight of this from that of the crude U_3O_8 .

Divide the filtrate into two portions. Test one for P_2O_5 , and if that is found deduct its amount from the crude U_3O_8 . Evaporate the other portion with H_2SO_4 , take up with water, and filter from $BaSO_4$, if present. If $BaSO_4$ is present, deduct its equivalent of BaO from the crude U_3O_8 . Next reduce the V_2O_5 accompanying the uranium by a current of SO_2 gas (not solution of SO_2) or by pure ammonium bisulphite, boil out SO_2 in a current of CO_2 , and titrate at 60° to 70° C. with $KMnO_4$. Deduct V_2O_5 so found from crude U_3O_8 and call the remainder U_3O_8 . Or, the second part of the nitrate solution may be evaporated with HCl to reduce V_2O_5 to V_2O_4 , then fumed with H_2SO_4 , filtered to remove $BaSO_4$ if present, and titrated for V_2O_4 as in the method for vanadium.

The fact that in the crude U_3O_8 the uranium may not be wholly U_3O_8 , but in small part UO_3 , does not invalidate the method for commercial needs.

Ledoux & Co., of New York, who have had much experience in the analysis of these ores, have furnished the details of the methods they use. They are as follows:

VOLUMETRIC METHOD FOR URANIUM IN CARNOTITE AND VANADIFEROUS URANIUM ORES.

The method given below is substantially Engle's method,^a but some changes in manipulation have been introduced which render it more accurate.

The method^b depends upon the separation of uranium as a phosphate from iron, vanadium, and other metals, the susceptibility of uranium phosphate to reduction by zinc and reoxidation by permanganate in a cold acid solution, and titration of the reduced uranium in sulphuric-acid solution by standard permanganate.

Many chemists who have experimented with the zinc-reduction method have laid stress on the difficulty of completely reducing uranium solutions and the danger of reoxidation by atmospheric oxygen during titration. It is our experience that reduction from UO_3 to UO_2 is easily attained, and further, that very prolonged reduction is apt to lead to formation of a lower oxide ($UO?$). This lower oxide is oxidized with great rapidity by atmospheric oxygen to the uranous condition, but solutions of uranous sulphate are stable in the presence of air and may be exposed to it with impunity. In fact, they may be agitated with air for several minutes without altering the state of oxidation in a measurable degree. The method is conducted as follows:

Treat 5 grams of the ore, or a quantity containing not over 0.3 gram of metallic uranium, in a No. 3 beaker with 10 c. c. of HNO_3 (1.42) and 20 c. c. of H_2SO_4 (1.84). Cover with a watch glass sup-

^a Engle, W. D., *Western Chem. and Met.*, Nov., 1908.

^b Pullman, O. S., jr., The determination of uranium and uranyl phosphate by the zinc reductor: *Am. Jour. Sci.*, 4th series, vol. 16, 1903, pp. 229-239; and Kern, E. F., The quantitative separation and determination of uranium, *Jour. Am. Chem. Soc.*, vol. 23, 1901, pp. 710-718.

ported above the beaker with glass hooks and evaporate till white fumes of H_2SO_4 begin to come off. Wash the cover with a little water, remove it, add a little more water to the beaker, mix the contents well and evaporate them almost to dryness, leaving a slightly moist residue containing about 2 c. c. of free H_2SO_4 . This operation is easily conducted on a good hot plate; it does not require much time if the temperature is sufficient and the draft is good.

The object of using a large excess of nitric and sulphuric acids is to destroy organic matter, which is present in many samples. The purpose of adding water after the first evaporation to fumes is to break up nitroso-sulphuric compounds and to completely eliminate nitric acid.

Add 75 to 80 c. c. of water to the residue and heat until all soluble matters are dissolved; then, without filtering, pass H_2S gas into the warm solution until all members of the H_2S group are precipitated. Filter and wash with warm water, collecting the filtrate and washings, which may measure 150 c. c., in a No. 3 beaker. Boil until all excess of H_2S is expelled and add H_2O_2 until all iron is oxidized. Neutralize the liquid with Na_2CO_3 and add about 2 grams in excess, then add 10 c. c. more of H_2O_2 and boil for 15 minutes. Filter through a 12.5-cm. S. and S. No. 489 paper and wash the precipitate four or five times with hot water, stirring it well with the jet of the wash bottle. Collect the filtrate and washings in a No. 5 beaker. Wash the precipitate from the filter into the No. 3 beaker with a fine stream of water, neglecting the small part that may adhere pertinaciously to the paper. Dissolve the precipitate in a little dilute H_2SO_4 , neutralize as before with Na_2CO_3 , adding about 2 grams in excess, add 10 c. c. of H_2O_2 and boil again as before. Filter through the same paper into the No. 5 beaker and wash the precipitate three times with hot water.

The combined filtrates and washings from the two Na_2CO_3 precipitations will contain all of the uranium and most of the vanadium. Add 5 grams of ammonium phosphate and 10 c. c. of H_2SO_4 to the liquid, boil it well until all CO_2 is expelled, make it slightly alkaline with ammonia and then slightly acid with acetic acid. Uranium is precipitated as ammonium-uranium phosphate and vanadium is retained in solution. Cool the beaker for three-quarters of an hour in ice water or let it stand over night. This precaution is necessary for the complete separation of the uranium precipitate. The precipitation appears to be more complete in faintly acid solutions than it is in ammoniacal liquids; hence the addition of acetic acid.

The precipitate is slimy and difficult to wash. Filter through a 12.5-cm. No. 589 S. and S. paper which has been treated with an emulsion of paper pulp. The paper pulp aids filtration and prevents the precipitate from passing through the paper.

Let the precipitate drain on the filter, and then from a wash bottle direct a stream of a dilute solution of ammonium sulphate (2 per cent) toward the inside seam of the filter paper at the top of the precipitate. The precipitate can thus be made to open affording a better drainage. Rinse the beaker with ammonium-sulphate solution and wash down the filter from the top until the paper is full. The precipitate is rather impervious and the object of this washing is to remove vanadium-bearing solution from the beaker and the filter paper rather than actually to wash the precipitate. After the filter has drained, remove as much as possible of the precipitate from it with a glass rod, transferring it to the beaker. Wash the filter well from the top to remove vanadium from it. While the filter is draining, add a little ammonium-sulphate solution to the beaker and beat the precipitate to a thin paste with it, breaking up all aggregations; then add about 100 c. c. of ammonium-sulphate wash-solution, stir well and filter as before. This method of washing is fairly rapid, and for large precipitates is more effective than washing on the filter would be; small precipitates may be washed on the filter.

Transfer the precipitate with a glass rod as completely as possible to the beaker and dissolve what remains on the paper in a little hot dilute (1:4) sulphuric acid, letting the acid run through the paper into the beaker containing the precipitate. Wash the paper well with hot water so that it may be used for filtering the second precipitation. See that all of the uranium phosphate in the beaker is dissolved in the acid. A little paper pulp will, of course, remain. Add 2 grams of ammonium phosphate, dilute to 250 c. c. with warm water, make slightly alkaline with ammonia, boil cautiously for several minutes and then make slightly acid with acetic acid. Cool the liquid as before and filter through the same filter, washing the beaker and the filter paper several times with ammonium sulphate solution. This procedure removes all or all but a small trace of vanadium in the case of the usual grade of uranium-vanadium ores, such as contain not over 4 per cent U_2O_5 and less than 6 per cent V_2O_5 . In the analysis of richer ores a third precipitation as phosphate may be required to remove every trace of vanadium.

Dissolve the uranium phosphate through the filter paper in hot sulphuric acid, using in all not over 40 c. c. of dilute (1:4) acid. Wash the filter well with water, let the solution cool and test it with a few drops of H_2O_2 to make sure that no vanadium is present. Since the solution is dissolved through the filter, the liquid should be free from any suspended paper pulp. Add an excess of a strong solution of $KMnO_4$, and heat the solution to boiling. The object of adding $KMnO_4$ is to destroy sugars and other organic compounds formed by the action of the solution on cellulose and traces of starch from the filter papers.

Omission of this precaution will often cause variations in the titration of solutions that have been passed repeatedly through filter paper.

Cool the liquid, dilute it to a volume of 150 c. c. and pass it through a reductor in the same way as in determining iron. The reductor we employ is simple.^a The zinc column is 19 cm. long and 2 cm. in diameter, of 30-mesh granulated zinc that has been amalgamated by treatment with a dilute solution of mercuric chloride. The reductor is worked by suction from a filter pump, and the precautions usual in making iron determinations are observed. The time required to pass the solution through the reductor is about 2 minutes, exclusive of washings.

After reduction, shake the solution vigorously for 1 minute with free access of air and titrate with a standard solution of KMnO_4 . Subtract 0.5 c. c. from the burette reading for an $\frac{N}{20}$ solution of permanganate as a correction for end point and reductor errors.

GRAVIMETRIC METHOD FOR VANADIUM AND URANIUM IN CARNOTITE AND OTHER ORES.

Treat from 2 to 5 grams of ore, according to the proportion of vanadium, iron, and uranium present, in a covered beaker, with 10 c. c. of HCl and let it stand 15 minutes, shaking it occasionally. Add 5 c. c. of HNO_3 and heat on a steam bath. When the solution is quiet, remove the cover and evaporate to dryness. Add 3 c. c. of HCl and 5 c. c. of water to the residue and let it stand on the steam bath for a few minutes, stirring occasionally. Dilute with 25 c. c. of hot water, filter into a small beaker, and wash the residue with warm water.

Some ores do not yield all the vanadium to this treatment, a little remaining with the insoluble residue. To make sure that all vanadium is in solution, ignite the residue in a platinum dish, treat it with 5 c. c. of HF , and evaporate to dryness on a steam bath. Do not bake the residue, for it is not necessary to expel all SiO_2 . Add 3 c. c. of HCl to the residue from the HF treatment and evaporate to dryness. Repeat this treatment to insure expulsion of HF . Treat the residue with 2 c. c. of HCl and 2 c. c. of water and stir with a glass rod until any red crust is dissolved, then dilute the solution with water and filter it into the main liquid.

Pass H_2S into the liquid to separate copper, lead, and other metals of this group, filter and boil the liquid to expel the H_2S . Concentrate the liquid to 100 c. c. if necessary, oxidize it with an excess of H_2O_2 , and then neutralize with dry Na_2CO_3 , adding 2 or 3 grams in excess. Boil the liquid for about 15 minutes until the yellowish uranium precipitate dissolves, leaving a brown precipitate which is

^a Blair, Analysis of iron and steel, 7th ed., p. 94.

principally iron. Filter and wash the iron precipitate with water, reserving the filtrate. Dissolve the iron precipitate in the least possible amount of HNO_3 (1:1), and add 10 c. c. of H_2O_2 , neutralize with Na_2CO_3 , add an excess of 2 grams of Na_2CO_3 , and boil as before. Filter into the beaker containing the first filtrate. The iron precipitate may contain a little vanadium—reserve it for further treatment.

Evaporate the united filtrates from the iron precipitation to a volume of about 200 c. c., add 10 c. c. of strong HNO_3 and boil until all CO_2 is expelled. Neutralize the free acid with ammonia (until a slight permanent precipitate appears), then add 4 c. c. of HNO_3 for each 100 c. c. of liquid. Now add 10 c. c. of a 20 per cent lead acetate solution, and enough of a strong solution of ammonium acetate to neutralize the nitric acid present and substitute acetic acid for it. The object is to precipitate the vanadium as lead vanadate in an acetic acid solution. The ammonium acetate solution may be made by mixing 80 c. c. of strong ammonia, 100 c. c. of water, and 70 c. c. of acetic acid 99 per cent pure.

Heat the liquid containing the lead vanadate precipitate on the steam bath for 1 hour or more, filter on a tight filter, and wash with warm water. Dissolve the precipitate in the least possible quantity of hot dilute nitric acid, neutralize as before, add 3 c. c. of nitric acid in excess, add 2 c. c. of lead acetate solution and repeat the precipitation of lead vanadate by adding ammonium acetate in excess, filter and add the filtrate to the one from the first precipitation of lead vanadate. Reserve the precipitate of lead vanadate for treatment as described below. Evaporate the united filtrates from the lead vanadate to about 400 c. c. Add 10 c. c. of strong H_2SO_4 to separate the bulk of the lead (derived from the excess of lead acetate) as PbSO_4 , filter and wash the precipitate with cold water. Neutralize the filtrate from the PbSO_4 with ammonia and add freshly prepared $(\text{NH}_4)\text{HS}$ until the solution is yellow and the uranium and what little lead is present are precipitated as sulphides. Warm the mixture on a steam bath until the sulphides settle well. Filter and wash slightly with warm water.

Dissolve the precipitate in a No. 2 beaker with hot dilute (1:2) HNO_3 , add 5 c. c. of H_2SO_4 and evaporate till fumes of H_2SO_4 appear, cool and take up with water, boil, and let the small precipitate of PbSO_4 settle until the solution is cold, filter the precipitate and wash it with very dilute H_2SO_4 .

SEPARATION OF ALUMINA.

Nearly neutralize the filtrate with ammonia, have the solutions cool (not warmer than 30°C.), and add powdered carbonate of ammonia in about 2 grams excess to precipitate the aluminum. Let the precipitate settle, filter, and wash it with warm water. If

the precipitate is bulky or is at all yellow, dissolve it in a little dilute H_2SO_4 and reprecipitate with carbonate of ammonia as described. Acidulate the filtrate from the alumina with H_2SO_4 , and boil thoroughly to expel CO_2 . Make the liquid slightly alkaline with NH_4OH while it is hot, and heat on the water bath until the ammonium uranate collects and settles. Filter and wash with a very dilute (2 per cent) solution of $(\text{NH}_4)\text{NO}_3$. Do not allow the precipitate to run dry on the filter after the first washing. Dry the precipitate, ignite it in a porcelain crucible, and weigh as U_3O_8 . Dissolve the precipitate in HNO_3 and test it with H_2O_2 for vanadium and with $(\text{NH}_4)_2\text{CO}_3$ for aluminum.

Dissolve the lead vanadate in dilute HNO_3 , add 10 c. c. of H_2SO_4 , and evaporate the mixture to fumes. Cool, take up with water, add 10 c. c. of a concentrated solution of SO_2 to the mixture, boil until the excess of SO_2 is expelled, and titrate the hot solution with a standard solution of potassium permanganate. The vanadium compound is reduced by SO_2 from V_2O_5 to V_2O_4 . The iron equivalent of the permanganate solution multiplied by 1.6329 = the V_2O_5 equivalent. It is not necessary to filter out the lead sulphate before boiling the mixture to expel the SO_2 . The boiling is best done in a large flask.

The iron precipitate that was produced by the addition of Na_2CO_3 and H_2O_2 to the original acid solution may contain vanadium. Ignite the precipitate in a platinum crucible and fuse the residue with Na_2CO_3 . Leach the fused mass with water, filter, and acidulate the filtrate with H_2SO_4 . The filtrate may be added to the main solution before reducing with SO_2 , or reduced and titrated separately, as preferred. In expelling the excess of SO_2 , it is necessary to boil the liquid for at least 10 minutes after the smell of SO_2 can no longer be detected.

Some of the factors used in calculations are: Fe value of permanganate times 0.9167 = V. Fe value of permanganate times 1.6329 = V_2O_5 . $1.78124\text{V} = \text{V}_2\text{O}_5$. $\text{U} = 0.84824\text{U}_3\text{O}_8$.

RAPID METHOD FOR THE DETERMINATION OF VANADIUM IN ORES IN THE PRESENCE OF IRON.

Treat 2 to 5 grams of the ore in a 16-ounce Erlenmeyer flask with 20 c. c. of HCl (sp. gr. 1.20) and warm for half an hour. Add 20 c. c. of water and 20 c. c. of H_2SO_4 and evaporate till fumes of H_2SO_4 are liberated. While the mixture is hot add powdered KMnO_4 , a little at a time, until all organic matter is oxidized and an excess of KMnO_4 is present. Heat a few minutes, cool, and add 25 c. c. of water and a few drops of a strong solution of KMnO_4 to insure complete oxidation. Add 50 c. c. of HCl (sp. gr. 1.20) and evaporate as rapidly as possible without causing "bumping" until the HCl is

expelled and H_2SO_4 fumes are evolved. Continue the fuming for 10 minutes. Cool, add a little cold water, dilute to 250 c. c. with boiling water, and determine the vanadium by titrating the hot solution with a standard solution of KMnO_4 .

In most cases it is unnecessary to remove the insoluble matter; sometimes, however, it is advisable to do so. Place the ore in a beaker, add 10 c. c. of HCl and 5 c. c. of H_2SO_4 , evaporate till fumes of H_2SO_4 are liberated, take up with water, and filter into the flask. Then add 15 c. c. more of H_2SO_4 and proceed as above.

If arsenic and molybdenum are present, they may be removed from the dilute H_2SO_4 solution before filtration by precipitation with H_2S .

Success with this method depends upon complete destruction of the organic matter and complete oxidation of the iron, etc., by KMnO_4 . Some vanadium ores contain much organic matter. In treating these the H_2SO_4 solution should be heated longer and several cautious additions of powdered KMnO_4 should be made. In some cases preliminary calcination at a low temperature will save time.

A glass rod placed in the Erlenmeyer flask lessens the tendency to "bump." The rod should be removed before the acid is completely evaporated.

The method depends on the reduction of vanadic compounds to divanadyl chloride by boiling with HCl and the conversion of this to divanadyl sulphate, while the other metals present remain fully oxidized.

The method of reduction by boiling with HCl was originated by Campagne, who states that reduction is incomplete if conducted in the presence of sulphuric acid, and therefore the acid should not be added until after the vanadium is completely reduced. We have found that in the presence of sufficient ferric iron the reduction by HCl is complete even in the presence of a large excess of H_2SO_4 . If the ores under analysis are low in iron and high in vanadium, iron must be added to insure correct results.

MINERALS OF URANIUM AND VANADIUM.

The classifications of the minerals of uranium and vanadium given in most works on mineralogy are incomplete. This incompleteness is due not only to a desire to condense such lists, but also to the fact that the commercial importance of uranium and vanadium compounds is of rather recent growth. For these reasons the authors have compiled from various sources a tabular description of the different minerals of uranium and vanadium. This classification, although necessarily incomplete in some respects, is given below for the use of those who may be interested.

Classification and description of the minerals of uranium and vanadium.

URANIUM MINERALS.

Name of mineral.	Composition.	General description and physical characteristics.	Content of metallic uranium.	Where found.
Ampangabeite. Autunite.....	$\text{Ca}(\text{UO}_2)_2(\text{PO}_4)_2 \cdot 8\text{H}_2\text{O}$, uranium and calcium phosphate.	Crystallization, orthorhombic; in plates or tabular crystals; pearly luster, or in micaceous aggregates; translucent; brittle. Cleavage, basal. Color, bright yellow. Hardness, 2 to 2.5. Specific gravity, 3.5 to 3.9.	<i>Per cent.</i> 17.091 49 -50.5	Madagascar. Connecticut, Massachusetts, Pennsylvania, North Carolina, South Carolina, Utah, France, Madagascar, Portugal, China, Cornwall, Austria, and Germany.
Betafite. Blomstrandite.....	Formula doubtful, tantalotitano columbate of uranium.	Amorphous.	24.66 24.1	Madagascar. Do.
Bröggerite, <i>see</i> Uraninite. Carnotite.....	$\text{K}_2\text{O} \cdot 2\text{UO}_3 \cdot \text{V}_2\text{O}_5 \cdot 3\text{H}_2\text{O}$ (?) uranyl potassium vanadate (?) containing barium and calcium.	Amorphous, perhaps slightly crystalline; pulverulent, in crusts intimately mixed with quartzose sand. Color, canary yellow. Specific gravity, 4.136.	52 -57	Colorado, Utah, and South Australia.
Chalcolite, <i>see</i> Torbernite. Cléveite.....	A variety of uraninite containing uranium, thorium, yttrium, and other rare earths as oxides (rich in helium).	Cubical crystals. Color, velvet black. Specific gravity, 7.49.	48 -61	Norway, Texas, South Carolina, Quebec, and Saxony.
Ebigite, <i>see</i> Uranothallite. Elasite, <i>see</i> Pittinite. Euxenite (polycrase).	$(\text{PbCaBa})\text{U}_2\text{SiO}_{12} \cdot 6\text{H}_2\text{O}$ (variety of gummitte). $\text{R}(\text{NbO}_2)_2\text{R}'(\text{TiO}_2)_3\text{H}_2\text{O}$, titano-columbate of yttrium, erbium, cerium, thorium, and uranium.	Orthorhombic, rarely in crystals, generally in masses. Color, brownish-black; when in powder form, yellowish to reddish brown. Hardness, 6.5. Specific gravity, 4.7 to 5.	5 -16	Adrianople, Turkey. Joachimsthal, Austria. Sweden, Norway, Ireland, North Carolina, and South Carolina.
Fergusonite (tyrite, bragite).	$\text{R}(\text{NbTa})\text{O}_4$, columbate and tantalate of yttrium, erbium, and other rare earths.	Tetragonal. Luster dull, on fractures vitreous and submetallic. Color, brownish-black; brittle. Hardness, 5.5 to 6. Specific gravity, 5.8.	2.1-8	Sweden, Norway, Greenland, Ceylon, Russia, Massachusetts, North and South Carolina, and Llano County, Tex.
Fritzscheite.....	$(\text{U}, \text{Mn}, \text{Va})\text{PO}_4 \cdot \text{H}_2\text{O}$	Square tables. Color, reddish-brown.		Bohemia and Saxony.
Gummite.....	$(\text{PbCaBa})\text{U}_2\text{SiO}_{12} \cdot 6\text{H}_2\text{O}$, alteration product of uraninite, often surrounding a black core of uraninite.	Amorphous, slightly crystalline. Color, reddish-yellow to orange-red and reddish-brown. Specific gravity, 3.9 to 4.2.	55 -60	Saxony, Bohemia, and North Carolina.
Hatchettite.....	$\text{R}(\text{NbTa})_2\text{O}_6 \cdot \text{H}_2\text{O}$, tantalotitanocolumbate of uranium.	In octahedrons. Color, yellowish brown. Specific gravity, 4.77 to 4.9.	12.5-13.5	Mitchell County, N. C.
Johannite.....	Formula doubtful; a hydrous sulphate of uranium and copper.	In druses or reniform masses; monoclinic. Color, apple green.	56	Bohemia and Saxony.
Kochellite, <i>see</i> Fergusonite. Lebigite.....	$\text{CaCO}_3(\text{UO}_2)\text{CO}_3 \cdot 20\text{H}_2\text{O}$, hydrous carbonate of uranium and calcium.	Transparent; luster vitreous on fracture. Color, apple green; occurs in thin coatings or mammillary concretions in uraninite.	31.5	Austria and Saxony.

Classification and description of the minerals of uranium and vanadium—Continued.

URANIUM MINERALS—Continued.

Name of mineral.	Composition.	General description and physical characteristics.	Content of metallic uranium.	Where found.
Mackintoshite.....	$\text{UO}_3\text{ThO}_3\text{SiO}_3\cdot 3\text{H}_2\text{O}$, silicate of uranium, thorium, cerium, and other rare earths. Alteration seems to yield thorogummite.	Tetragonal in square prisms with pyramids; commonly massive, nodular. Fracture small, subconchoidal; luster, dull. Color, black; opaque. Hardness, 5.5. Specific gravity, 4.38 to 5.	Per cent. 20	Llano County, Tex.
Medjидite.....	Sulphate of uranium and calcium.	Transparent; slightly crystalline.		Adrianople, Turkey.
Microлите.....	Pyro-tantalate of calcium, tungsten, tin, manganese, beryllium, yttrium, cerium, and other rare earths.	Regular octahedrons in brown, brownish, and reddish crystals. Hardness, 5.5. Specific gravity, 5.49 to 6.13.	1.6	Virginia, Connecticut, Massachusetts, Greenland, Sweden, and Elba.
Nivenite, <i>see</i> Uraninite. Nohlite, <i>see</i> Samarskite. Phosphuranylite...	$(\text{UO}_2)_2\cdot \text{P}_2\text{O}_5\cdot 6\text{H}_2\text{O}$, uranyl phosphate.....	Crystalline, pulverulent, incrustations of deep lemon-yellow color.	60 -64	Mitchell County, N. C.
Pilbarite.....	A hydrated silicate of lead, uranium, and thorium.	Color, bright canary-yellow. Hardness, 2.5 to 3. Specific gravity, 4.4 to 4.7.		Pilbarra, Goldfield, Australia in tantalite lode.
Pitchblende, <i>see</i> Uraninite. Polycrase, <i>see</i> Euxenite.	Columbate and titanite of yttrium, cerium, erbium, uranium.	Orthorhombic; prismatic or tabular crystals; luster, vitreous to resinous. Color, black or brownish. Hardness, 5 to 6. Specific gravity, 4.97 to 5.04.	66.3-64.6	Sweden, Norway, and North and South Carolina.
Priorite, <i>see</i> Blomstrandite. Randite.....	$(\text{UO}_2, \text{UO}_3, \text{ThO}_3)$	Yellow incrustations in granite.		Swaziland, Africa.
Rowlandite.....	Carbonate of uranium?..	Massive with gadolinite. Color, drab-green.		Pennsylvania.
Rutherfordin.....	Silicate of yttrium.....	Yellow crystals.....	3- .5	Llano County, Tex.
Rutherfordordite.....	Uranyl carbonate of lead, etc.		69.5	German Africa.
Rutherfordordite, variety of fergusonite.	Titanite of cerium.....	Monoclinic crystals. Color, brownish-black.	68	Rutherford County, N. C.
Samarskite.....	$\text{R}_2\text{R}_3(\text{NbTa})_2\text{O}_{21}$, tantalocolumbate of yttrium, erbium, calcium, uranium, thorium, iron.	Orthorhombic crystals; massive; fracture, conchoidal; luster, vitreous to resinous. Color, velvety black. Hardness, 5 to 6. Specific gravity, 5.6 to 5.8.	3 -14	Sweden, Norway, North Carolina, Colorado, and Canada.
Samiresite.....			18.67	Madagascar.
Schroëckingerite.....	Carbonate of uranium and sulphate of calcium.	Orthorhombic; various yellow colors.		Joachimstahl, Austria.
Thorogummite.....	$\text{UO}_3\text{ThO}_3\text{SiO}_3\cdot 6\text{H}_2\text{O}$, silicate of uranium, thorium, lead, etc.; occurs with fergusonite, etc.	Color, brownish black.....	18	Llano County, Tex.
Thorianite.....	$\text{ThO}_2\cdot \text{U}_3\text{O}_8$, oxide of uranium, thorium, and other rare earths.	Black cubic crystals.....	4 -10	Ceylon (Gambola and Sabaragamuva).
Thorite (uranothorite).	ThSiO_4 , anhydrous silicate of thorium.	Tetragonal and also massive. Hardness, 4.5 to 5. Specific gravity, 4.8 to 5.4.	1 -19	New York and Norway.
Torbernite.....	$\text{Cu}(\text{UO}_2)_2\cdot \text{P}_2\text{O}_8\cdot 8\text{H}_2\text{O}$, Copper uranite.	Tetragonal crystals, foliated; micaceous; brittle. Luster, pearly; subadamantine. Transparent to translucent. Color, emerald green and grass-green, some specimens apple or siskin green. Hardness, 2 to 2.5. Specific gravity, 3.4 to 3.6.	47 -51.4	Cornwall, Saxony, and Austria.

a In crystals.

b Massive.

Classification and description of the minerals of uranium and vanadium—Continued.

URANIUM MINERALS—Continued.

Name of mineral.	Composition.	General description and physical characteristics.	Content of metallic uranium.	Where found.
			<i>Per cent.</i>	
Troegeerite	$(\text{UO}_2)_2\text{As}_2\text{O}_5 \cdot 12\text{H}_2\text{O}$	Monoclinic crystals in druses of tabular crystals. Color, lemon yellow.	56.2	Saxony.
Tyuyamunite	$\text{CaO}(\text{NO}_3)_2 \cdot \text{V}_2\text{O}_5 \cdot x\text{H}_2\text{O}$..	Calcium carnotite; small yellow crystals.	54	Ferghana, Russian Turkestan; Colorado, Utah.
Uranaitite	$\text{UO}_2\text{UO}_2\text{PbO}$, etc., oxide of uranium with rare earths, Pb, Ca, Fe, Bi, Mn, Cu, Si, Al.	Rarely in octahedral crystals; usually massive. Luster, sub metallic, dull. Fracture, conchoidal; brittle. Color, black to dark brown, olive green, grayish; opaque. Hardness, 5.5. Specific gravity, 9 to 9.7 in crystals, or 6.4 and higher when massive.	65-80	Connecticut, North Carolina, Texas, Central City, Colo., South Dakota, Ottawa (Quebec), Saxony, Austria-Hungary, Norway, Spain, and East Africa.
Uranochalcite, <i>see</i> Uranopillite.				
Uranocircite	$\text{Ba}(\text{UO}_2)_2 \cdot \text{P}_2\text{O}_5 \cdot 8\text{H}_2\text{O}$	Orthorhombic	47	Germany.
Uranophane	$\text{CaO}(\text{UO}_2)_2(\text{BiO}_2)_2 \cdot 6\text{H}_2\text{O}$, Pb, Ba.	In radiated aggregations; massive; fibrous. Color, yellow. Specific gravity, 3.81 to 3.9.	43-56	Silesia, Bavaria, North Carolina, and Spain.
Uranopillite	Uranyl-calcium sulphate	Crystalline. Color, yellow in velvety incrustations.	64	Saxony.
Uranosphærite	$(\text{BiO})_2\text{U}_2\text{O}_7 \cdot 3\text{H}_2\text{O}$	In hairlike globular, and aggregate forms. Color, orange yellow to brick red.	42	Saxony.
Uranospinitite	$\text{Ca}(\text{UO}_2)_2\text{As}_2\text{O}_5 \cdot 8\text{H}_2\text{O}$	In tabular orthorhombic crystals rectangular in outline. Color, siskin green.	49	Saxony and Utah.
Uranothallite (flu-therite).	$2\text{CaCO}_3\text{U}(\text{CO}_3)_2 \cdot 10\text{H}_2\text{O}$, occurs on pitchblende.	In scaly or granular crystalline aggregations; orthorhombic small crystals. Color, siskin green.	31-32	Austria.
Uranothorite, <i>see</i> Thorite.				
Voglianite, uncertain uranium sulphates from Joachimsthal, Austria, <i>see</i> Uranopillite.				
Walpurgite	$\text{Bi}_{10}(\text{UO}_2)_8(\text{OH})_{28}(\text{AsO}_4)_4$	Triclinic, in thin yellow crystals resembling gypsum. Specific gravity, 5.76.	16.5	Saxony.
Yttrocrasite	Titanate of yttrium, thorium, and uranium (complex).	Color, yellow to black	2.2	Burnet County, Tex.
Yttrotantalite	Tantalate and columbate of yttrium, erbium, iron and uranium, tungsten and tin.	Luster, submetallic to vitreous and greasy. Color, black, brown, brownish yellow, straw yellow; opaque, subtranslucent.	.5-3.7	Sweden.
Xenotime	YPO_4 , essentially yttrium phosphate (also cerium, iron, manganese, calcium).	In crystals resembling sircon in habit; in rolled grains; brittle. Luster, resinous to vitreous. Color, yellowish brown, reddish brown, flesh red, grayish white, wine yellow, pale yellow. Hardness, 4 to 5. Specific gravity, 4.45 to 4.56.	2.9	Norway.
Zeunerite	$\text{Cu}(\text{UO}_2)_2\text{As}_2\text{O}_5 \cdot 8\text{H}_2\text{O}$, uranyl-copper arsenate.	In tabular crystals, tetragonal, resembling torbernite in form and color. Specific gravity, 3.2.	50-53	Saxony, Austria, and Cornwall.
Zippelite, <i>see</i> Voglianite.				

The following minerals also contain uranium: Ainnerodite, 13 per cent; davidite (Olari, South Australia); hielmite, 4.1 per cent; naegite (silicate); a mineral found in Japan), 23 per cent; sipylite (complex niobate), 3.3 per cent; uraconite (uranocher, *see* uranopillite); vetinghofite, *see* samarskite; voglite (complex carbonate), 30.7 per cent.

Classification and description of the minerals of uranium and vanadium—Continued.

VANADIUM MINERALS.

Name of mineral.	Composition.	General description and physical characteristics.	Content of vanadium, as V_2O_5 .	Where found.
Albite.	$V_2O_5 \cdot H_2O$.		<i>Per cent.</i> 90.83	Siberia.
Ardennite (dewalquite).	Vanado-silicate of aluminum and manganese; also contains arsenic.	Prismatic crystals resembling ilvaite. Color, yellow to brownish-yellow. Hardness, 6 to 7. Specific gravity, 3.62.	9	Ardennes, Belgium.
Brackebuschite (near descloizite).			24-25	Argentina.
Calcium vanadate.		Radiated aggregations; pulverulent. Color, claret.		Colorado and Utah.
Calciovolborthite.	$(CuCa)_2V_2O_7(CuCa)(OH)_2$, basic vanadate of copper and calcium.	In thin green tables; also gray, finely crystalline, granular.	37-39	Colorado and Thuringia, Germany.
Carnotite.	$K_2O \cdot 2UO_3 \cdot V_2O_5 \cdot 3H_2O$, uranyl-potassium vanadate (?) containing barium and calcium.	A morphous, perhaps slightly crystalline, pulverulent incrustations; intimately mixed with quartzose sand. Color, canary yellow. Specific gravity, 4.136.	19-21	Colorado, Utah, and South Australia.
Chileite, <i>see</i> Mottramite.			13-14	
Cuprodescloizite.	$(PbCu)_2(OH)VO_4$ (also called tritochorite, ramirite).		17-22	Colorado.
Dechenite, <i>see</i> Descloizite.	PbV_2O_6 .	Massive; botryoidal; nodular. Color, deep red to yellowish-red and brownish red. Specific gravity, 5.6 to 5.81.	45-47	Rhenish Bavaria.
Descloizite.	$(PbZn)_2(OH)VO_4$, basic vanadate of lead, zinc, etc.	In small crystals; also massive, fibrous, radiated with mammillary surface. Color, cherry-red and brownish-red black. Hardness, 3.5. Specific gravity, 5.9 to 6.2.	20-22	Argentina, Carinthia, New Mexico, and Arizona.
Endlichite.	Variety of vanadinite with arsenic.			
Eusynchite, <i>see</i> Descloizite.			17-21	
Fritzcheite.	Compounds of uranium, manganese, vanadium, phosphorus.	Square tables, reddish-brown.		Autun (France), Bohemia, and Saxony.
Mottramite.	Variety of cuprodescloizite.	Velvety black incrustations.	17-18	Cheshire, England.
Patronite.	Sulphide of vanadium (iron, sulphur, silica, molybdenum, nickel, etc.).	Black to grayish-black.	34	Minasragra, Peru.
Psittacinite.	Variety of cuprodescloizite.	Thin coatings; also pulverulent. Color, siskin to olive green.	17-26	Montana.
Pucherite.	$BiVO_4$, bismuth vanadate.	In small orthorhombic crystals. Color, reddish-brown. Hardness, 4. Specific gravity, 6.249.	22-27	Schneeberg, Saxony.
Roscoelite.	$H_2K_2(MgFe)(Al,V)_4(SiO_3)_{12}(?)$, vanadium mica.	In minute scales; structure micaceous. Color, brown to greenish brown and greenish. Specific gravity, 2.92 to 2.94.	21-29	California, Colorado, and Utah.
Sulvanite.	Cu_2VS_4 .		24.76	South Australia.
Turanite.	$V_2O_5 \cdot 5CuO \cdot 2H_2O$.		29.36	Siberia.

Classification and description of the minerals of uranium and vanadium—Continued.

VANADIUM MINERALS—Continued.

Name of mineral.	Composition.	General description and physical characteristics.	Content of vanadium, as V_2O_5 .	Where found.
Vanadic ochre..... Vanadinite.....	$(PbCl)Pb_2(VO_4)_2$, chloro-vanadate of lead.	Hexagonal-pyramidal crystals, prismatic, often hollow prisms, in globules and incrustations; fracture uneven to flat, conchoidal; brittle. Color, deep ruby-red, light-brownish, yellow and reddish-brown; subtranslucent to opaque. Hardness, 2.75 to 8. Specific gravity, 6.66 to 7.10.	Per cent. (?) 19.4	New Mexico, Arizona, Ural (Russia), Carinthia, Sweden, and Argentina.
Vanadinite.....	Altered cuprodesclizite.		44-45	
Volborthite.....	$(Cu Ca Ba)_2(OH)_2VO_4 \cdot 6H_2O(?)$, hydrous vanadate of copper, barium, and calcium.	In small six-sided tables and in globular form. Color, olive-green, lemon-yellow.	14-15	Ural (Russia), and Colorado.

PUBLICATIONS ON MINERAL TECHNOLOGY AND METHODS OF MINING.

The following Bureau of Mines publications may be obtained free by applying to the Director, Bureau of Mines, Washington, D. C.:

BULLETIN 10. The use of permissible explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls., 4 figs.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 44. First national mine-safety demonstration, Pittsburgh, Pa., October 30 and 31, 1911, by H. M. Wilson and A. H. Fay, with a chapter on the explosion at the experimental mine, by G. S. Rice. 1912. 75 pp., 7 pls., 4 figs.

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INDEX.

A.	Page.		Page.
Adams, Orr J., work of.....	9	Carnotite ore region, map showing.....	10
Alsdorf, G. W., work of.....	9	Cassel, Ernest, work of.....	62
Alumina, separation of, from carnotite ores..	89, 90	Caywood claims (Colo.), carnotite deposits at, description of.....	10, 11
Annie May claim (Colo.), carnotite deposits at, description of.....	28	Cherrington, C. S., work of.....	9
Arnold, J. O., work of.....	57	Cliff mine (Colo.), carnotite deposits in, description of.....	25, 26
Australia, uranium deposits in, description of.	50, 51	view of.....	24, 26
Austria, pitchblende deposits in, description of.....	48, 49	Cloud claim (Colo.), carnotite deposits at, description of.....	29
B.			
Beck, Richard, work of.....	48	Club Ranch, Colo., carnotite deposits at, description of.....	24, 25
Belcher mine (Colo.), pitchblende deposits in, description of.....	45	Coal Creek, Colo., carnotite deposits near, description of.....	10, 11
Black Fox claim (Colo.), carnotite deposits at, description of.....	28, 29	Concentration of ores, advantages of.....	35, 36
Blackbourne claim (Colo.), carnotite deposits in, description of.....	27	cost of.....	39, 40
Bleeker, W. F., process of, for recovery of vanadium, description of.....	70, 71, 78	factors in.....	39
Bob-o-link claim (Colo.), carnotite deposits at, description of.....	29	dry, methods of.....	38
Bobtail claim (Colo.), carnotite deposits at, description of.....	28	tests of, results of.....	39
Boltwood, B. B., work of.....	66, 68	necessity of.....	35, 36
Boot Leg claim (Colo.), carnotite deposits at, description of.....	29	possibility of.....	35, 40
Boutwell, J. M., work of.....	14	wet, method of.....	36
Brown, H. C., work of.....	9	tests of, results of.....	37
Brown, H. L. Y., work of.....	50	Cripple Creek claim (Colo.) carnotite deposits at, description of.....	23
"Bug hole," definition of.....	20	Cross, —, work of.....	51
description of.....	30, 31	Cumenge, —, work of.....	18
Bull Canyon, Colo., carnotite deposits near, description of.....	28, 29	Cummings, W. L., work of.....	9
C.			
Calhoun mine (Colo.), pitchblende deposits in, description of.....	46	Cummings claims (Colo.), carnotite deposits at, description of.....	28, 29
California, vanadium deposits in, description of.....	54	Curran, T. V., work of.....	9
Cameron, Angus, work of.....	9	Curran claims (Colo.), carnotite deposits at, description of.....	23
Carnotite, chemical treatment of, possibility of.....	41	Cutter, N. Mex., vanadium deposits at, description of.....	53
concentration of, necessity of.....	35, 36	D.	
possibility of.....	35	Dry concentration. <i>See</i> Concentration.	
deposits of, description of.....	9-29	E.	
extent of.....	41, 42	Eagle, Colo., vanadium deposits near, description of.....	53, 54
extraction of radium from.....	77	East Paradox Valley, Colo., carnotite deposits at, description of.....	26-28
in Austria, extent of.....	42	Electroscope, description of.....	64
in United States, importance of.....	42	figure showing.....	66
mining of, cost of.....	32	method of using.....	64
methods of.....	32	standardization of, correction for.....	68
occurrence of.....	36	method of.....	66, 67
origin of.....	30	use of, in determination of ores.....	64
separation of alumina from.....	89, 90	precautions necessary in.....	64
transportation of, cost of.....	33, 34	view of.....	26
difficulties of.....	33	Elkhorn claims (Colo.), carnotite deposits at, description of.....	10, 11
treatment of, with nitric or hydrochloric acid.....	76, 77	Engle, W. D., work of.....	72, 85
F.			
Fawn claim (Colo.), carnotite deposits at, description of.....	29		
Fischer, Siegfried, process of, for recovery of vanadium.....	73		

	Page.		L.	Page.
Fleck, Herman, work of.....	9, 19, 51, 74	Last Chance claim (Colo.), carnotite deposits at, description of.....		28
Fleck, method for extraction of uranium and vanadium, description of.....	74	Little Emma claim (Colo.), carnotite deposits at, description of.....		12
Forsman claim (Utah), carnotite deposits at, description of.....	14	Little Ruth claim (Utah), carnotite deposits at, description of.....		15
French, S. W., work of.....	51	Little Tom claim (Colo.), carnotite deposits at, description of.....		28
Friedel, —, work of.....	18	Logan, Thomas, work of.....		18
G.		Long Park, Colo., carnotite deposits at, analyses of.....		20
Gale, H. S., work of.....	10	description of.....		21-24
German mine (Colo.), pitchblende deposits in, description of.....	45	Lookout claim (Colo.), carnotite deposits at, description of.....		12
Germany, pitchblende deposits in, description of.....	48, 49	Lorimer claim (Utah), carnotite deposits at, description of.....		14, 15
Gilpin County, Colo., pitchblende deposits in, description of.....	43-46	Loveless-Forsman claim (Utah), carnotite deposits at, description of.....		15
Grahamite, analysis of.....	55	M.		
deposits of.....	54	McBride, James, work of.....		18
description of.....	54	McIntyre district, Colo., carnotite deposits at, description of.....		29
Grand View claim (Utah), carnotite deposits at, description of.....	15	McKeever claim (Colo.), carnotite deposits at, description of.....		26
Green River, Utah, carnotite deposits near, description of.....	13-16	Meeker, Colo., carnotite deposits near, description of.....		10, 11
Greenback claims (Colo.), carnotite deposits at, description of.....	27	Melrose Discovery claim (Utah), carnotite deposits at, description of.....		15
Greystone claim (Colo.), carnotite deposits at, description of.....	28	Mitchell County, N. C., pitchblende deposits in, description of.....		43
H.		Monayunk claim (Utah), carnotite deposits at, description of.....		15
Haitinger, Ludwig, work of.....	79	Monogram claim (Colo.), carnotite deposits at, description of.....		27
Haldane, W. G., work of.....	19, 51	N.		
Hall, Henry, work of.....	9	Napoleon claim (Utah), carnotite deposits at, description of.....		15
Happy Thought claim (Colo.), carnotite deposits at, description of.....	28	Nevada, grahamite deposits in, description of.....		54
Haynes, J. H., work of.....	72	Newmire, Colo., vanadium deposits near, description of.....		51, 52
Head, George A., work of.....	9	Nitric acid, use of, in treatment of ores.....		76, 77
Hequembourg, K. D., work of.....	9	North Star mine (Colo.), carnotite deposits in, view of.....		16
Hess, F. L., work of.....	14, 19, 51, 53	O.		
Hillebrand, W. F., work of.....	19, 21, 29, 30, 51, 52, 82, 84	Oklahoma, grahamite deposits in, description of.....		54
Holliday claim (Colo.), carnotite deposits at, description of.....	28	Opera Box claim (Colo.), carnotite deposits at, description of.....		27
Huerfano County, Colo., vanadium deposits in, description of.....	52, 53	P.		
Humery, —, work of.....	49	Paradox Valley, Colo., carnotite deposits at, description of.....		19-29
Hydraulic, Colo., carnotite deposits at, description of.....	29	map of.....		22
I.		Patronite, deposits of, in Peru.....		55
Iveagh, —, work of.....	62	description of.....		55, 56, 95
J.		Pitchblende, concentration of, possibility of.....		47
Jacobs claim (Colo.), carnotite deposits at, description of.....	26	deposits of, description of.....		43-47
Jasper claim (Colo.), carnotite deposits at, description of.....	28	method of treating.....		79, 80
Jo Dandy claim (Colo.), carnotite deposits in, description of.....	27	wastes in mining of.....		46, 47
K.		Placerville, Colo., vanadium deposits near, description of.....		51, 52
Kent Smith claim (Colo.), carnotite deposits at, description of.....	27	Portugal, uranium deposits in, description of.....		49, 50
Kimball, Gordon, work of.....	18	Poulot, —, work of.....		18
Kirk mine (Colo.), pitchblende deposits in, description of.....	44	Purinton, —, work of.....		51
Koenig, G. A., process of, for recovery of vanadium.....	73			
Kuppelweiser, —, work of.....	61			

Q.	Page.		Page.
Quarter Circle claim (Colo.), carnotite deposits at, description of.....	27	Thunderbolt mine (Colo.), carnotite deposits in, description of.....	26
R.		Turner claims (Colo.), carnotite deposits at, description of.....	21-23
Radclyff, Sidney, method of, for recovery of radium.....	75	U.	
Radioactivity of sandstone. <i>See</i> Sandstone, radioactivity of.		Ulrich, Karl, work of.....	79
Radioactivity of water, variations in.....	59	Uraninite, separation of emanation from, apparatus for, figure showing.....	67
Radium, determination of.....	66	Uranium, deposits of, in Australia, description of.....	50, 51
method for.....	68, 69	in Portugal, description of.....	49, 50
extraction of, from carnotite ore.....	77	determination of, difficulties in making.....	82
from pitchblende.....	79, 80	methods for.....	82-86, 88, 89
importance of.....	70	origin of.....	30
method of.....	75	percentage of, in ore.....	34
misuse of, in treatment of disease.....	61	recovery of, method of.....	72
price of.....	62	uses of.....	58
production of, in foreign countries.....	57	Uranium minerals, list of.....	92-95
in United States.....	56	Uranium oxide, percentage of, method of calculating.....	65
sale of, fraud practiced in.....	62	production of.....	56
use of, in treatment of disease.....	58, 59, 63	selling price of.....	34, 35
apparatus for.....	59	V.	
results of.....	60	Valley View claim (Colo.), carnotite deposits at, description of.....	27
Radium Institute, English, work of.....	62	Vanadic acid, extraction of, from copper vanadate.....	78
Radium Institute of Vienna, scope of.....	61	Vanadium, determination of, difficulties in making.....	82
Ransome, F. L., work of.....	19, 21, 29, 30, 51, 52	method for.....	82-84, 88, 90, 91
Richardson, Clifford, work of.....	54	effect of, on steel.....	57
Rickard, Forbes, work of.....	9, 45	origin of.....	30
Rutherford, E., work of.....	66	percentage of, in ore.....	34
S.		production of, in United States.....	57
San Miguel County, Colo., vanadium deposits in, description of.....	51, 52	recovery of, method of.....	70-74
San Raphael River, Utah, carnotite deposits near, description of.....	14	separation of, from uranium.....	78
Sandstone, radioactivity of.....	31	uses of.....	57, 58
Saucer Basin, Colo., carnotite deposits at, description of.....	25, 26	Vanadium minerals, list of.....	92, 95
Saunders claim (Colo.), carnotite deposits at, descriptions of.....	29	Vanadium oxide, selling price of.....	34, 35
Segaud, —, work of.....	49	Vernon Junior claim (Utah), carnotite deposits at, description of.....	15
Skull Creek Basin, Colo., carnotite deposits in, description of.....	11, 12	Voilleque, —, work of.....	18
Spencer, —, work of.....	51	W.	
Stewart, Newt., work of.....	9	Wardvern claim (Utah), carnotite deposits at, description of.....	15
Sundown claim (Colo.), carnotite deposits at, description of.....	29	Wedding Bell claim (Colo.), carnotite deposits at, description of.....	29
Swindler mine (Colo.), carnotite deposits at, description of.....	23	West Virginia, grahamite deposits in, description of.....	54
view of.....	24	Wet concentration. <i>See</i> Concentration.	
T.		White, I. C., work of.....	54
Table Mountain, Utah, carnotite deposits at, description of.....	16	Whittemore, Charles F., work of.....	9
Taff, J. A., work of.....	54	Widow claim (Colo.), carnotite deposits at, description of.....	29
Talbert, Andrew J., work of.....	18	Willmarth, O. B., work of.....	9
Taylor, David, work of.....	9	Wilson claims (Colo.), carnotite deposits at, description of.....	25, 26, 28
Telluride No. 8 claim (Utah), carnotite deposits at, description of.....	17	Wood mine (Colo.), pitchblende deposits in, description of.....	46
Thompsons district, Utah, carnotite deposits at, description of.....	16-18		
view of.....	16		

APPENDIX.

By reason of the widespread public interest in radium and its use in the cure of cancer and other malignant diseases, it has been deemed advisable to present in this revised edition of Bulletin 70 such additional information concerning the mining, treatment, and utilization of radium, uranium, and vanadium as seems most important of that collected since the material for the first edition was compiled.

MINING OF RADIUM-BEARING ORES DURING 1913.

CARNOTITE.

During 1913 Paradox Valley and the surrounding districts in the southwestern part of Colorado were the scenes of the greatest activity since the opening of the carnotite deposits. Most of the mining was done in the districts that were producing ore toward the close of 1912. There was considerable activity at Long Park, Club Ranch, Hydraulic, and Bull Canyon. Two fields that were not developed in 1912 were opened, one at and around Gateway, which yielded one shipment of a few tons of ore; and another in the western part of Paradox Valley, which was the original scene of the main operations. The McIntyre district during the latter part of 1913 was not very active. Some prospecting was done toward the end of the year, mostly on Carpenters Flats and on Lyon Creek, a branch of La Sal Creek. Both of these localities are at the west end of Paradox Valley; there seems to be a tendency in prospecting to work toward the deposits opened a number of years ago.

At present the largest owners of claims are Standard Chemical Co. 140, Thos. F. V. Curran and associates 81, General Vanadium Co., of Liverpool, 58, Radium Co. of America 22, Colorado Carnotite Co. 23, Crucible Steel Mining & Milling Co. 16, W. L. Cummings 22. Other owners have only a few claims each.

In Utah, at Green River, not nearly as much mining was done as in 1912, because the Radium Co. of America's claims were worked to a small extent, the company having much ore left in the bins from 1912. Only assessment work was done in the Thompsons district, but there was considerable activity near Moab, especially to the southeast, in the neighborhood of Pack Creek. Several carloads of ore were shipped by James Wade. This ore was hauled to Thompsons, the rate for the haul being about \$15 a ton.

A new district in Utah has been opened about 7 miles northeast of Monticello. No ore was shipped from this locality during 1913 but

a number of claims were staked. H. H. Redd, E. Whitney, and J. Ennis have 23 claims. Other owners in this district are W. W. and J. N. Wright. It is hoped that although the haul to a railroad is more than 75 miles, the cost will not be more than \$25 or \$30 per ton by sending out ore on the return trip of freighters.

Another new district has been opened in the Henry Mountains, about 100 miles southeast of Green River. Little development work has yet been done there and no shipments have been made.

Two miles southeast of Fruita, in the southern part of Utah, is a deposit carrying low-grade copper and uranium, the latter in the form of sulphate. The uranium streak is from 1 to 2 inches wide and has no commercial importance.

No ore was shipped during 1913 from Coal Creek, near Meeker, Colo., from Skull Creek, Colo., or from Table Mountain, Utah.

PITCHBLENDE.

There was much more activity at pitchblende mines during 1913 than in 1912. At the end of 1912 the German and Belcher mines were taken over by the German & Belcher Mines Co., a corporation controlled by Alfred I. DuPont, of Wilmington, Del. During 1913 the mining was done in a systematic manner, resulting in the largest output obtained from these mines in many years, if not the largest that has ever been obtained. During the year several tons of high-grade pitchblende were produced in addition to a considerable quantity of low-grade milling ore. The Wood mine, which was taken over by the DuPont interests during the year, produced a few hundred pounds of high-grade ore. The Calhoun mine, the ownership of which did not change, produced 1,000 pounds of 30 per cent pitchblende, nearly all of which was sold for specimens. The Kirk mine was not worked in 1913, but was unwatered for examination by prospective purchasers from abroad.

One shipment of about 50 tons of low-grade ore from the Kirk mine was sold abroad during the year. It assayed 1.49 per cent U_3O_8 and brought \$1.25 per pound U_3O_8 f. o. b. Denver. This ore was mined several years ago. All the ore mined from claims controlled by Mr. DuPont has been held for possible treatment in this country.

SHIPMENTS OF CARNOTITE ORES.

The Standard Chemical Co., of Pittsburgh, Pa., the main producer of radium in the United States, made its first shipments of ore in April. Toward the end of the year, especially in November and December, its shipments were rather heavy, as it was then shipping ore that had been mined during the summer. The other shipments consisted largely of ore mined in 1912 and stored at Coke

Ovens, in the Paradox Valley. The General Vanadium Co., of Liverpool, England, which has been one of the heaviest producers, worked its claims only about six months in the year and its shipments in 1913 were little more than half those of 1912. Other shipments from the Paradox were made by T. F. V. Curran and associates, A. M. Wilson, the Colorado Carnotite Co., W. L. Cummings, J. Hart, Crucible Steel Mining & Milling Co., and J. M. Belisle.

In Utah the Radium Co. of America and W. Anderson shipped ore from Green River and James Wade shipped ore from Moab.

According to advance figures issued by the United States Geological Survey,^a the output of uranium and radium ores was nearly 50 per cent larger in 1913 than in 1912, the total output being 2,140 tons of dry ore carrying an equivalent of 38 tons uranium oxide. Of this uranium oxide 19.25 tons, containing an equivalent of 7.4 grams radium bromide, seem to have been shipped to Europe, and 18.74 tons, containing an equivalent of 7.2 grams anhydrous radium bromide, were retained in this country. Although the tonnage of ore retained in this country was larger than that shipped to Europe, a greater amount of uranium oxide was actually exported, as it was the richer ores that went abroad.

The carnotite ores remaining in this country are to be treated by three firms—the Standard Chemical Co., of Pittsburgh, Pa.; the Radium Co. of America, at Sellersville, Pa.; and the National Radium Institute, at its plant at Denver, Colo.

The estimates made above are on the basis of an equilibrium ratio of uranium to radium in carnotite ores of 90 per cent of the theoretical—a figure which has been shown to be true of most of the ores examined at the Denver laboratory of the Bureau of Mines. In 1912 the production was 11.43 grams radium bromide. The value of the radium contained in the ore mined in 1913, at the rate of \$120,000 per gram of metallic radium, would approximate \$1,050,000.

As already stated, the General Vanadium Co. produced less during 1913. The plant owned by the American Rare Metals Co., in the McIntyre district, south of the Paradox, was in operation to some extent in the early part of the year, but since that time has been shut down. The larger part of the radium concentrates produced by them in 1912 and the early part of 1913 were sold abroad in December, 1913. There was a decided tendency on the part of some operators to limit production, with the object of either awaiting better prices or of holding the ore until methods of extraction are published so that companies may be formed for the manufacture of radium in this country. This tendency, combined with the small output of the General Vanadium Co., made the amount of ore shipped abroad less than it would

^a Press Bulletin 152, January, 1914.

otherwise have been; but notwithstanding this decrease, although only about half of the total ore shipped went abroad, the actual amount exported was very little less than that exported in 1912, the increased production being used in this country.

PRODUCTION OF RADIUM SALTS.

A number of small companies have started work in different ways during the year. The statements in the public press concerning these companies have not always been correct. For example, one company was advertised as putting up a plant near Monticello, Utah, for the extraction of radium. In reality all that this company plans at the present time is to concentrate low-grade ores and sell the concentrates. Its capital is small and it has no present intention of going into the manufacture of radium salts. Again, statements regarding the Radium Bank of Grand Junction, Colo., might give the impression that this organization has a considerable amount of radium for loan or rent. In fact, the amount that the bank possesses must be very small, for only a few hundred pounds of high-grade ore has been treated. Other companies have been advertised, but in the early part of the year 1914 there were no companies actually at work or in a position to begin the commercial extraction of radium other than the Standard Chemical Co., the Radium Co. of America, the National Radium Institute, and the American Rare Metals Co. The latter company made a radium concentrate, but does not refine its product.

It is difficult to ascertain the total production of radium preparations from ores other than American.

The mines belonging to the Austrian Government produced, in 1912, 1,698 grams of metallic radium, the equivalent of 2.89 grams of anhydrous radium bromide. This production was smaller than in 1911, although special efforts were made to increase the output. The figures for 1913 are not yet available. Apparently these mines are not producing as much as formerly.

The Cornwall, England, pitchblende mines seem to be doing better, and the last report of the British Radium Corporation (Ltd.) states that it is putting up a larger plant with increased capacity and that the prospects for future development are very good.

In South Australia the Mount Paynter ores have been sold in England, something over 100 tons, including both high-grade and low-grade material, having been thus exported. The ore is chiefly autunite. The output from the Radium Hill deposits near Olary is shipped to Sydney, New South Wales, and the radium extracted there. The ore, which consists of uraniferous ilmenite mixed with carnotite, is concentrated and the concentrates are treated chemically. So far the production of radium has probably been small.

The autunite deposits of Portugal have been worked steadily, but the material is low grade and, on account of its occurrence in hard igneous rock, is difficult to concentrate by mechanical means. The ore is treated chemically in Portugal and some ore is shipped to France and England.

Mexico has produced some pitchblende. At one mine, in Yucatan, 160 miles west of Progreso, the pitchblende is said to occur in nodules, some of which contain gold and have the equilibrium amount of radium, and others not containing gold have less than the equilibrium amount of radium. This statement has not been confirmed. The ore is found in mica-schist and granite and is shipped to France. Two other small deposits of pitchblende have been found in northern Mexico in the Province of Chihuahua. One of these deposits, southwest of El Paso, is opened by a shaft 70 feet deep with two levels. The other deposit is southeast of El Paso and almost due west of Marfa. No shipments were made in 1913.

During 1913 the total production of radium from ores other than American carnotite was probably between 6.5 and 7.5 grams of radium bromide, an increase over 1912. Notwithstanding this increase, American ores once more yielded over two-thirds of the world's production of radium. Owing to the great demand for radium at the present time, foreign companies are planning to increase their output decidedly during 1914. Some companies, using either pitchblende or autunite, are expecting to produce as much as 2 or 3 grams of radium bromide per annum, and the Austrian Government hopes to increase its production. Undoubtedly the total amount produced from ores other than American carnotite will be larger during 1914 than in 1913 and, therefore, any decrease in shipments of ore from this country will not necessarily work any great hardship to European buyers.

It is interesting to note that half of the total supply of radium in existence at the present time has been furnished by American ores during the last three years.^a However, although the carnotite deposits of the United States are the largest known radium-bearing ore deposits in the world, they will not last indefinitely. Exaggerated statements have appeared in the newspapers and exaggerated estimates of the amount of material available have been published. It is extremely important that every effort should be made to eliminate waste in the mining and treatment of these ores so that both the United States and the world at large can get the maximum amount of benefit from them. There is a tendency to use better methods in mining and to keep the low-grade, nonshipping ore separate from waste material, and undoubtedly in 1914 serious attempts will be made to concentrate ores that are too low-grade to ship.

^a This statement refers to the radium in all the ores mined to January, 1914. Naturally all of this radium has not yet been extracted.

PRICES FOR URANIUM ORES.

The price of carnotite ore showed a decided upward tendency during 1913. This rise began about May and greatly increased during the year. In 1912, 2 per cent carnotite brought \$2 per pound of U_3O_8 f. o. b. New York or Europe. At present, February, 1914, the same ore can be sold in Placerville, Colo., for \$2.10 per pound of U_3O_8 , an increase of nearly 20 per cent, as under the old arrangement the shipper paid the freight to New York or Europe. Several lots have been purchased at this figure. An offer of \$2.25 a pound for a carload of 2.2 per cent ore was made in Denver and refused. Other agents offer \$82.50 per ton for $2\frac{1}{2}$ per cent ore; \$110 a ton for 3 per cent ore, and \$148 per ton for 4 per cent ore f. o. b. Placerville, Colo. The prices abroad are considerably higher than are paid to the miners, but the agent has to pay freight to Europe, insurance, and run the risk that the check assays do not correspond. All grades of ore from 1 per cent up can now be sold, whereas in 1912 it was impossible to sell anything carrying under 2 per cent U_3O_8 . It is doubtful, however, whether even at the increased price the shipment of ore carrying less than $1\frac{1}{2}$ per cent U_3O_8 pays, except possibly from Green River and Thompsons, Utah, the wagon hauls from these points being much shorter than those from other places. The miners who can ship carloads directly to Europe, without selling through American agents, can, of course, get a 10 or 15 per cent increase over the figures given above.

The ore is still sold on its content of uranium oxide and not on its radium content. In only one case to the writer's knowledge has the radium content been made the basis of sale, but as a definite ratio between the radium and uranium was specified in the offer, the price was really based on the uranium oxide content.

There seems to be a decided tendency to pay better prices for high-grade ore than in 1912, the price per pound U_3O_8 increasing rapidly for the richer grades of ore.

It is difficult to ascertain the exact price at which pitchblende can be sold, foreign agents refusing to make quotations except when an actual sale is under consideration, and as the amount of pitchblende sold during the past year has been very small, it may be stated that there is no definite market price for the mineral at the present time. In general, however, the price follows very closely that of carnotite, and although pitchblende contains a little more radium than carnotite, this fact is perhaps fully counterbalanced by the latter mineral being more easily treated. One offer made by an agent for pitchblende was \$3 per pound U_3O_8 for a 20 per cent ore f. o. b. Denver.

PRICES FOR FINISHED PRODUCTS.

The price of radium has increased during the past year. During the summer the price rose to \$120,000 a gram of metal and since

November radium has not been purchasable for immediate delivery in appreciable amounts anywhere in this country or abroad. Future deliveries are offered at the rate of \$120,000 a gram of radium metal in salts of 60 per cent concentration to \$180,000 a gram of metal in salts of 90 to 95 per cent purity. Probably such prices will not hold, as they are out of all proportion to the cost of production. Speculation is rife and dealers are taking full advantage of the increased demand and the limited supply. Practically all of the factories, both abroad and in this country, have advance orders that will take care of their production for at least six months, and, in the case of some, for a year. This statement also applies to the four or five foreign companies that are manufacturing mesothorium, which is being used to a considerable extent as a substitute for radium in medical work. One hundred and twenty thousand dollars a gram radium metal represents a price of \$91,000 a gram for anhydrous radium chloride and \$70,000 a gram for anhydrous radium bromide.

One unfortunate phase of the commercial exploitation of radium is the fact that a considerable proportion of the radium produced is going into low-grade preparations. This is due to the fact that the price obtained for the radium in such preparations is higher than even the highest figures at which it can now be sold as a purified product. Some manufacturers are therefore catering specially to this demand. Radioactive bath salts, waters, salves, and muds are offered, although the value of such use is questionable. The unfortunate feature of such preparations, however, is that the radium is permanently lost and the amount of material that would be available for the treatment of cancer is therefore materially reduced. One foreign company deliberately leaves some of its radium in certain of its precipitates and residues, these being sold as fertilizer, radioactive muds, etc.

USES FOR RADIUM AND URANIUM.

RADIUM.

Radium is used to a small extent in making luminous paint for use on watches and clocks so that they can be seen in the dark. Its main use, however, is medical, and radium therapy has been developed considerably during the past year, especially in connection with the treatment of cancer. In treating cancer the alpha and beta rays are shut off by incasing a tube of radium or its emanation in a small lead, silver, or platinum capsule. The radium can then be applied externally to the cancer or an incision made and the capsule inserted. The length of exposure depends upon the size of the cancer and the quantity of radioactive material available. Some decidedly remarkable cures have been reported. The consensus of opinion among well-known physicians who have had experience with radium treatment

appears to be that success depends largely upon the amount of material available, large dosage being recommended, the use of gamma rays from one-half gram to 1 gram being now not unusual. Wickham ^a advises heavy dosage and long exposure.

Exner, ^b Kroenig, Gauss, ^c and Schindler ^d all agree with Wickham and recommend heavy dosage. The old method was to use a small amount of radium and not filter the alpha and beta rays from the gamma rays. In this way burns were frequent and there was little or no beneficial effect on the cancer. It is only since the new method of using the gamma rays only has become prevalent that positive results have really been achieved. Kelly ^e states that radium is now being used to supplant surgery in external cancers, especially in those about the face, where it frequently effects a painless cure. It appears to be especially effective in cancer of the larynx, of the womb, in some breast cancers, in pruritus, in lupus, and in vascular growths and moles, etc. Present difficulties lie in dealing with the whole group of breast cancers and of internal growths in the chest and abdomen, although even here some successes have been noted. Kelly seems to think that the failures are mainly due to lack of material rather than to the particular type of the disease.

The radium emanation treatment is still being used for arthritis, rheumatism, and other similar diseases.

URANIUM.

The main use for uranium is still as a coloring agent in the manufacture of glass and pottery. Ferrouanium is much more difficult to make than ferrovandium, but a German firm has recently advertised the sale of ferrouanium in commercial quantities.

OPERATING CONCERNS.

The Radium Chemical Co., of Pittsburgh, Pa., allied with the Standard Chemical Co., has had radium preparations for sale since the summer of 1913. This radium is prepared from the carnotite ores obtained from the claims of the Standard Chemical Co. in the Paradox Valley, Colo.

The Radium Co. of America, Sellersville, Pa., also manufactured radium salts during 1913, and has stated that it would have some radium for sale about January 1, 1914. This company owns and operates claims at Green River, Utah.

The American Rare Metals Co., which has had a mill for two years in the McIntyre district south of the Paradox Valley, was at work

^a Deutsch. Med. Wochenschr., 1913, p. 391.

^b Wiener Klin. Wochenschr., 1913.

^c Muenschener Med. Wochenschr., 1913.

^d Wiener Klin. Wochenschr., 1913.

^e Kelly, H. A., Abbe, Robert, and Burnam, C. F., Hearings before House Committee on Mines and Mining, Jan. 19, 1914, pp. 1-34.

early in 1913, but shut down about the middle of the year. This company makes a crude radium concentrate and does not refine any of its products.

The National Radium Institute was incorporated in 1913, with Howard A. Kelly, of Baltimore, and James Douglas, of New York, as its principal directors. The institute has obtained the right to mine a number of claims in Long Park, near the Paradox Valley, Colo. Under an agreement with the Bureau of Mines, the technical operations of the mines and mill are to be guided by the scientific staff of the bureau. A plant has been erected in Denver, Colo., where the best methods for treating carnotite ore will be thoroughly tested. Concentration experiments will be conducted in the field and if successful will be applied to reducing wastes that now take place. After the preliminary experiments, operations will be continued on a larger scale. The institute will study also the separation of uranium and vanadium. All processes, details of apparatus and plant, and general information gained will be published for the benefit of the people. The institute has been formed for the special purpose of securing enough radium to conduct extensive experiments in radium therapy with special reference to the cure of cancer; it also expects to carry on experiments regarding the physical characteristics and chemical effects of radium rays.

The activities of the institute are sure to be of benefit to the prospector and miner by providing a greater demand for ore; to the plant operator by developing methods and by creating a larger market for his product; and to the people by aiding the treatment, and possibly the cure, of the most malignant of diseases.

The radium produced is intended for the institute's own use and will not be for sale.

CONCENTRATION OF LOW-GRADE ORES.

Various persons have attempted to concentrate the low-grade carnotite ores, which during 1913 were more carefully separated by the miners from the barren rock. One company in Utah near Monticello is now installing some machinery to crush the rock and obtain a concentrate by sifting. Although the carnotite particles can be concentrated in this way and a slight enrichment can be made, it is necessary to subject the crushed material to attrition in order to liberate the fine powdery substance from the quartz grains.

In most places where carnotite is mined, little if any water is obtainable, except at an excessive cost for transportation or piping, and it is doubtful whether hauling the low-grade ore to the river or other places where water can be found will prove profitable at present prices.

The National Radium Institute is making experiments with dry concentration and the method, if found efficient, will be applied on a

larger scale at its claims at Long Park and Hydraulic, Colo. The system proposed is similar to that outlined on page 68 of this bulletin. An especially designed attrition apparatus is now being constructed which, if found successful, can be duplicated without charge by all other operators in the field and can be used in connection with other apparatus necessary for efficient concentration. All data in regard to the concentration experiments made by the National Radium Institute will be published later.

As yet, little more can be done to prevent waste than to encourage miners to separate low-grade milling ore from waste rock and to save all the material obtained by hand sorting the shipping ore.

SAMPLING CARNOTITE ORES.

Much care should be taken in sampling carnotite ores on account of the possible loss of dust rich in uranium. In the past, ore usually has been hand sampled by the old method of taking a shovelful from each bag of ore, or a shovelful from a series of bags. These shovelfuls of ore were then crushed to small size, thoroughly mixed, and formed into a cone, this cone being quartered down and the final sample being taken with split samplers.

Ore shipped to Denver by the National Radium Institute was treated in the plant of Sutton, Steele & Steele Co. of Denver. The ore was crushed to one-half inch size, run through a mechanical drier, and then elevated to pass, first, an 8-mesh and then a 40-mesh screen. The oversize passed through rolls and the product was sifted through the 40-mesh screen. All apparatus was as nearly air-tight as possible and was connected with dust collectors, the dust being collected separately in bags. At the end of the 40-mesh screen was an automatic sampler through which all the screened material passed, the cut-out set to receive the sample across the stream of ore once in every 27 seconds. The sample obtained was thoroughly mixed, formed into a cone, quartered, and further reduced by means of a split sampler. The ore as received had an average moisture content of about 3 per cent, which was reduced by drying to an average of 1.4 per cent in the final product.

Much dust was collected by the dust collectors showing how much valuable mineral will be lost in sampling if the apparatus used is not air-tight, and if the dust is not collected. If proper precautions are used, the loss in dust can be made very small.

PRODUCTION OF VANADIUM ORE.

In this country during 1913 the vanadium ores produced were almost entirely from the roscoelite mines of the Primos Chemical Co., near Newmire, Colo. There was no production from other claims containing roscoelite ores in the vicinity of the Newmire plant. The product in the form of iron vanadate is shipped to Primos, Pa.,

where it is converted into ferrovanadium. Of the vanadium in carnotite ores a little over one-half of the production was shipped to Europe in the ore. No reports have been received of any vanadium being produced from other vanadium-bearing ores mined in this country, such as vanadinite.

As in 1912, most of the ferrovanadium produced in this country was derived from patronite mined and shipped from Minasagra, Peru, South America.

OTHER DEPOSITS OF VANADIUM ORE.^a

Near Bayard, Grant County, N. Mex., a station on a branch of the Atchison, Topeka & Santa Fe Railway, is the Lucky Bill lead-vanadium mine. The property has been described by Larsh.^b The mineral is vanadinite, or vanadate of lead. Here, as in many other places in New Mexico and Arizona, the lead-vanadate is either thrown on the dump or shipped with the other lead ores to the smelter. The vanadium content of the rock is usually so small that it is doubtful if it would pay at present prices to treat the mineral especially for vanadium. In the smelting of the vanadinite ore the vanadium might be recovered from the slag, but the difficulties would be considerable.

There was no production of vanadium at the Lucky Bill mine during 1913. There are a number of other places in the lead-copper districts of New Mexico and Arizona where vanadinite is found, but the ore is usually even lower grade than that of the Lucky Bill mine.

One other occurrence of vanadium ore, which may be important, is at the Shattuck-Arizona mine at Bisbee, Ariz. At the 600-foot level, a short distance from the shaft, is a large vug, or cavity, lined with a vanadium mineral, probably cuprodesclowitzite. The foot-wall is limestone, the hanging wall brecciated quartz, and between them along a fault is the large vuglike deposit. The vug has caved in in places and it is therefore difficult to give its dimensions and the extent of the vanadium ore. However, there seems to be enough ore in sight to justify saving it. The company is considering the possibility of commercially extracting the vanadium from the ore on a small scale.

The mineral itself is described by Wells,^c who says:

The mineral occurs in the form of stalactites. The smaller aggregates radiate from a narrow base and end in rounded clusters $\frac{1}{2}$ mm. in diameter. The larger growths occur in reniform masses several centimeters in diameter. The latter are coated with a red powder, but the smaller aggregates have an olive hue.

Although the physical character of the mineral would lead one to believe that it is cuprodesclowitzite, the chemical analysis shows some

^a Not mentioned in previous discussion.

^b Larsh, P. A., Lucky Bill lead-vanadium mine. Eng. and Min. Jour., vol. 96, Dec. 13, 1913, p. 1103-1104.

^c Wells, R. C., A new occurrence of cuprodesclowitzite. Am. Jour. Sci., December, 1913, p. 636-638.

difference from the composition of the mineral as known heretofore. It is rather difficult to say where the line is to be drawn between mottramite, psittacinite, and cuprodescloizite. At certain places in the deposit there is a mineral of a different crystalline structure, consisting of lustrous black, velvety incrustations. The crust is thin, and an analysis [by C. F. Whittemore] indicated that its composition is nearly the same as that of cuprodescloizite. The whole deposit is high in vanadium.

There was no activity in mining vanadium ores in the Sangre de Christo Range in Huerfano County, Colo.; at Cutter, N. Mex.; or at any places in New Mexico or Arizona where vanadinite was reported. About 70 miles north of Tucson, Ariz., near Mammoth, are veins being worked for gold, lead, and some copper where vanadinite is mixed in many places with the wulfenite that is found throughout the veins. Although there seems to be a good quantity of wulfenite available, the vanadinite is of minor importance, although it may be obtained as a by-product by careful sorting. The company which is working the old Mammoth mine is expecting to separate the wulfenite if a suitable market can be found for the mineral or concentrates. The mine was examined by a Pittsburgh concern for its content of vanadium, but evidently the amount found was too small to justify operation for vanadium.

Nearby are the Collins and Mohawk mines which also show considerable wulfenite and some vanadinite. The Mohawk is not being worked at present.

Vanadinite is also found at Hillsboro, Cooks Peak, Chloride, and Georgetown, and near Silver City, in New Mexico, and at many points in Arizona, but nowhere in quantities sufficient to justify mining at present prices.

The deposit of vanadinite 5 miles from Klinefelter Station, near Needles, Cal., which shows vanadinite intimately mixed with small wulfenite crystals is not opened extensively. No work was done there during the year, though the owner expects to develop the deposit in the hope of finding better ore at depth.

MARKET AND PRICES.

As nearly all of the ferrovanadium in this country was produced from patronite imported from Peru, no prices for vanadium ore could be obtained.

There was no change in the price quoted for ferrovanadium during the year 1913, it being \$2.25 per pound of vanadium, as in 1912.

USES OF VANADINIUM.

No new uses for vanadium have been reported. The main use is as an alloy in steel as already stated.

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12

Bulletin 72

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
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OCCURRENCE OF EXPLOSIVE GASES IN COAL MINES

BY

N. H. DARTON

[The investigations of the occurrence of gas in coal mines in southern Illinois were conducted under a cooperative agreement with the department of mining engineering of the University of Illinois and the Illinois State Geological Survey]



WASHINGTON
GOVERNMENT PRINTING OFFICE
1915

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CONTENTS.

	Page.
Introduction.....	11
The gases in coal and the conditions controlling their escape.....	12
General considerations.....	12
Character of mine gases.....	13
Methane.....	13
Carbon dioxide.....	13
Carbon monoxide.....	14
Nitrogen and oxygen.....	14
Fire damp.....	14
Nature of gases in coal beds.....	15
Composition of blower gas.....	16
Gas from English blowers.....	16
Gas from Austrian mines.....	17
Gas from Prussian mines.....	17
Gas from bore holes in anthracite region, Pennsylvania.....	18
Gas from French mines.....	19
Conditions under which gas is given off by coal.....	19
Occurrence of "blowers" or "feeders".....	20
Occurrence of gas outbursts.....	21
Causes.....	21
Instances of serious outbursts.....	21
Three Pennsylvania mines.....	21
Two Belgian mines.....	22
Morissey, B. C.....	22
Abercarn mine, Wales.....	22
Crushed coal a cause of gas outbursts.....	22
Discussion of outbursts in Belgian coal fields.....	23
Outbursts in Agrppe, Produits, and Marcinelle-Nord mines.....	23
Investigations of Stassart and Lemaire.....	24
Outbursts due to roof fall or movement of face.....	25
Gas from squeezes.....	26
Methane in coal.....	26
Origin.....	26
Relation to class of coal and metamorphism.....	27
Condition of methane in coal.....	28
Relation of methane to age of coal.....	29
Relation of methane to depth of coal.....	29
Relation of methane to structural features.....	30
Relation of methane to character of cover.....	31
Diffusion of methane.....	31
Gases in coal samples.....	32
Gas from samples of English coals.....	32
Gas from samples of German coals.....	33
Gas from samples of Belgian coals.....	34
Gas from samples of American coals.....	35

The gases in coal and the conditions controlling their escape—Continued.	Page.
Rate of escape of gas from coal.....	36
Effect of crushing.....	36
Rate of escape from lump coal.....	37
Rate of escape from pulverized coal.....	39
Comparative tests of different sizes.....	40
Method of testing.....	40
Conclusions.....	41
Variation in volume from different coals.....	42
Tests of various American coals.....	42
Method of testing.....	42
Results of tests.....	43
Tests of Illinois coals.....	45
Effect of oxidation.....	46
Amount of methane in mine air.....	48
Factors affecting volume.....	48
Percentages in Prussian mines.....	49
Percentages in Austrian mines.....	50
Percentages in Russian mines.....	51
Percentages in mines of Scotland.....	51
Relation of volume of methane to amount of coal mined or exposed.....	52
Investigation in Saarbrücken basin, Germany.....	52
Emanation from mines near Wilkes-Barre, Pa.....	52
Emanation from mines in Saxony.....	52
Results of Austrian investigation.....	53
Emanation from Prussian mines.....	54
Emanation from a mine at Liévin, France.....	56
Daily variations in methane emanation.....	58
Winkler's investigations.....	58
Tests of Prussian commission.....	58
Tests by Hilt.....	58
Tests by Liveing.....	60
Pressure of gas in coal beds.....	60
Gas pressure in coal beds in England.....	61
Method of conducting tests.....	61
Results obtained.....	62
Mallard's view of Wood's observations.....	63
Gas pressure in coal beds in Wales.....	64
Gas pressure in coal beds in Belgium.....	64
Investigations of Schorn, Watteyne, and Maquet.....	64
Tests by Maquet.....	67
Conclusions from results of pressure tests in Belgium.....	69
Gas pressure in coal in Liévin mine, France.....	69
Tests in Frederic bed.....	70
Tests in Alfred bed.....	70
Tests in Leonard bed.....	71
Gas pressures in coal at St. Etienne.....	72
Results of tests.....	72
Conclusions.....	73
Gas pressure in coal in Austrian mines.....	73
Volume of gas from holes in solid coal.....	73
Tests at St. Etienne.....	73
Tests at Liévin.....	74
Tests by Wood in English mines.....	74
Effect of rock and water pressure on gas pressure in coal.....	75

The gases in coal and the conditions controlling their escape—Continued.	Page.
Permeability of rocks and coal to gas.....	76
Escape of gas from strata near Wilkes-Barre, Pa.....	76
Escape of air from sealed chamber in Maryland mine.....	77
Gas in strata adjoining the coal.....	78
Gas in a sandstone stratum in Pennsylvania.....	79
Relation of volume of gas to underground temperature.....	80
Coal oxidation as a source of gas.....	80
Increase in temperature from oxidation.....	80
Absorption of oxygen by coal samples.....	81
Effects of variation of atmospheric pressure on gas emanation.....	81
Investigations in British mines.....	83
Investigations in Prussian mines.....	84
Tests by Hilt near Aix-la-Chapelle and Karwin.....	84
Review of Hilt's work by Mallard and Le Chatelier.....	86
Investigations in Austrian mines.....	86
Tests by Koehler at Karwin, Aachen, and Hibernia.....	86
Conclusions by Behrens.....	88
Tests at Hanover mine.....	89
Investigations of the Austrian Fire Damp Commission.....	90
Investigations in French mines.....	90
Tests at Liévin.....	91
Atmospheric conditions in old workings.....	92
Relation of barometric pressure to amount of gas from fresh coal surfaces.....	94
Effect of settling of roof and floor.....	95
Relative volumes from different workings under various conditions.....	95
Conclusions derived from results of tests at Liévin.....	97
Observations in England and the United States.....	97
Tests with samples of Pennsylvania coal.....	98
Effects of earth movements on methane emanation.....	98
Duration of methane emanation from coal.....	101
Gas drainage in mining.....	101
Relation of methane content to character of coal.....	102
Probable error in figures for methane content in mine air.....	103
Explosive gas in the northern anthracite coal field of Pennsylvania.....	104
Introduction.....	104
General structure of the northern anthracite basin.....	106
Collection of mine-air samples.....	107
Conditions in representative mines.....	108
Lance mine.....	108
Succession and structure of beds.....	108
Return-air samples.....	109
Red Ash returns.....	110
Ross returns.....	111
Bennett returns.....	112
Cooper returns.....	114
Five-foot returns.....	115
Hillman returns.....	116
Résumé of conditions in Lance mine.....	117
Methane outflow from Hillman and Five-foot beds.....	118
Outflow from Cooper and Bennett beds.....	118
Outflow from Ross and Red Ash beds.....	119
Generalizations as to methane emanations.....	119
Influence of river deposits.....	121

Explosive gas in the northern anthracite coal field of Pennsylvania—Continued.	
Conditions in representative mines—Continued.	
	Page.
South Wilkes-Barre mine.....	122
Return-air samples.....	122
Air samples from the Abbott returns.....	123
Air samples from the Hillman returns.....	125
Résumé of conditions in South Wilkes-Barre mine.....	126
Dorrance mine.....	127
Succession and structure of beds.....	128
Return-air samples.....	128
Air samples from Red Ash returns.....	129
Composition of Red Ash coal.....	130
Air samples from Baltimore returns.....	131
Air samples from Five-foot returns.....	132
Air samples from Hillman returns.....	132
Résumé of conditions in Dorrance mine.....	133
Hoyt mine.....	134
Air samples from Red Ash returns.....	134
Résumé of conditions in Red Ash workings.....	136
Mine No. 14, near Pittston.....	137
Air samples.....	137
Air samples from Pittston returns.....	138
Air sample from Checker return.....	138
Composition of Pittston coal.....	138
Sloan mine.....	139
Air samples from Dunmore returns.....	139
Structure of Dunmore bed.....	141
Character of Dunmore coal.....	142
Amount of methane in relation to area of coal exposed.....	142
Carbon dioxide in mine air.....	143
Chance of leakage of gas to higher workings.....	143
Effect of cessation of mining on mine air.....	144
Results of sampling air of idle mines.....	144
Discussion of results.....	146
Lance mine.....	146
South Wilkes-Barre mine.....	146
Dorrance mine.....	146
Hoyt mine.....	147
Sloan mine.....	147
Résumé.....	147
Miscellaneous observations in mines in the anthracite field.....	147
Mines in and about Scranton.....	147
Bellevue and Pancoast mines.....	148
Marvine mine.....	148
Dickson mine.....	148
Green Ridge and Diamond mines.....	148
Storrs, Cayuga, and Brisbin mines.....	149
Manville and Grassy Island mines.....	149
Mines between Pittston and Hudson.....	150
Mine No. 6.....	150
Pine Ridge mine.....	150
Delaware mine.....	152
Mines south of Wilkes-Barre.....	153
Baltimore No. 5 mine.....	153
Cunnyngham mine.....	154

Explosive gas in the northern anthracite coal field of Pennsylvania—Contd.	
Miscellaneous observations in mines in the anthracite field—Continued.	
Mines south of Wilkes-Barre—Continued.	Page.
Hollenback mine.....	154
Maxwell No. 20 mine.....	156
Mines near Nanticoke and Hanover.....	157
Auchincloss mine.....	157
Bliss mine.....	159
Truesdale mine.....	161
Mines between Luzerne and Avondale.....	163
Harry E. mine.....	163
Pettebone mine.....	164
Kingston No. 4 mine.....	165
Plymouth No. 2 mine.....	166
Dodson mine.....	167
Nottingham mine.....	168
Avondale mine.....	169
Methane from culm.....	169
Pressure of gas in the anthracite field.....	170
Gas in bore holes in the anthracite field.....	170
Gas conditions in anthracite field compared with those in other American mines.....	172
Some features of fire-damp explosions in the northern anthracite field...	173
Explosive gas in mines in the southern part of the Illinois coal field.....	175
Introduction.....	175
General features of the coal measures in southern Illinois.....	175
Distribution and relation of the coal beds.....	175
Stratigraphy.....	176
Structure.....	177
Character of the coals.....	178
Occurrence of gas in southern Illinois mines.....	179
Conditions in representative mines in Herrin bed.....	180
Mines near Benton.....	180
Hart-Williams mine.....	180
Benton mine.....	181
Hanaford mine.....	182
Mines near Christopher.....	183
United mine.....	183
Zeigler district mine.....	183
Zeigler mine.....	184
Mines near Duquoin.....	185
Paradise mine.....	186
Majestic mine.....	187
Davis mine.....	187
Horn mine.....	187
Mines near West Frankfort.....	188
Dering No. 11 mine.....	188
Dering No. 18 mine.....	190
Wilmington-Star mine.....	190
Mines about Johnston City.....	191
Standard or East mine.....	191
West mine.....	192
Blackbrier mine.....	193
New Virginia mine.....	193

Explosive gas in mines in the southern part of the Illinois coal fields—Contd.

Occurrence of gas in southern Illinois mines—Continued.

Conditions in representative mines in Herrin bed—Continued.

	Page.
Mines about Johnston City—Continued.	
Carterville district mine.....	194
Oak Ridge mine.....	194
Mines northeast of Marion.....	194
Colp mine.....	195
Scranton mine.....	195
West Virginia mine.....	195
Mines northwest of Marion.....	195
Peabody No. 3 mine.....	195
Peabody No. 2 mine.....	196
Big Muddy mine.....	196
Mines south and west of Herrin.....	197
Hemlock mine.....	197
Taylor No. 2 mine.....	198
Taylor No. 1 mine.....	199
Sunnyside mine.....	199
Chicago & Carterville Co. mines.....	199
"A" mine.....	200
"B" mine.....	200
Pond Creek, Possum Ridge, and Rend No. 2 mines.....	201
Pond Creek mine.....	201
Possum Ridge mine.....	202
Rend No. 2 mine.....	202
Mines north of Carterville.....	202
Burr C mine.....	203
Madison No. 8 mine.....	203
Madison No. 9 mine.....	203
C. & B. M. mine.....	204
Mines No. 7 and No. 8 of the Big Muddy Coal & Iron Co.....	204
Bush and Royaltown mines.....	205
Hallidayboro mine.....	206
Mine at Galatia.....	207
Conditions in representative mines in Harrisburg bed.....	207
Mines about Harrisburg.....	207
O'Gara No. 15 mine.....	207
O'Gara No. 7 mine.....	208
O'Gara No. 9 mine.....	208
O'Gara No. 3 mine.....	208
O'Gara No. 12 mine.....	209
O'Gara No. 1 mine.....	210
Mines near Eldorado.....	210
O'Gara No. 10 mine.....	210
O'Gara No. 8 mine.....	211
Nigger Hill mine.....	211
Conditions in representative mines in Murphysboro bed.....	212
Mines near Murphysboro.....	212
No. 9 mine.....	212
Harrison mine.....	212
Blair No. 1 mine.....	213

Explosive gas in mines in the southern part of the Illinois coal field—Contd.

Occurrence of gas in southern Illinois mines—Continued.

Pressure of gas in coal beds.....	213
Method of making test.....	213

ERRATA.

Page 14, 2d line from top. After the word "combustion" insert "Some physiologists say that large amounts of CO₂ have toxic effects other than suffocation."

8th line from top. After the words "coal dust" insert "It may be found that CO never occurs naturally in coal but is a secondary product."

18th line from bottom. After the word "gas" insert "The nitrogen is nearly, if not all, from the air in the mine."

12th line from bottom. Add "Air containing even considerably more than 7 per cent of oxygen might not support life, but the limit depends on many conditions."

6th line from bottom. For "10 to 30 per cent" read "5 to 30 per cent."

Page 20, 18th line from bottom. For "discharged a volume estimated at 47,000 feet" read "discharged for a short time a volume estimated at 47,000 feet."

Page 59, first table. Volumes of methane are cubic feet per minute, not cubic meters per minute.

Page 77, 10th line from top. Add "The term 'gaseous' in a strictly classificatory sense is now often applied to any mine containing sufficient gas to cause an accident under ordinary conditions."

Page 87. The barograph record in figure 11 is not reversed as stated.

Page 103, 13th and 14th lines from top. For "error of at least 1 per cent" read "error much greater than 1 per cent."

8th line from bottom. For "0.10 per cent" read "10 per cent."

6th line from bottom. For "0.10 per cent" read "10 per cent."

Page 105, 2d line above table. For "31,000 cubic feet of methane a minute, or 5,760,000 cubic feet a day, weighing 130 tons" read "31,000 cubic feet of methane a minute or 43,640,000 cubic feet a day, weighing nearly 1,000 tons."

93749°—15

conditions.....	40
3. Diagram showing rate of escape and volume of methane from 8 samples of American coals.....	43

	Page.
FIGURE 4. Gas pressure in holes of various depths in English mines.....	63
5. Plan and sections showing location of holes for gas-pressure tests at Belle-Vue mine, Belgium.....	65
6. Diagram of pressures in test holes at Belle-Vue mine.....	66
7. Diagram of pressures in test holes in coal at Belle-Vue mine, Belgium, while the rock tunnel was being continued to the coal....	66
8. Diagram showing position of test holes at Beaulieuart mine and points at which gas outbursts occurred in driving the rise.....	67
9. Diagram of pressures in test borings at Beaulieuart mine.....	68
10. Curves showing variations in volume of methane and carbon dioxide in relation to variations in atmospheric pressure, Gemeinschaft and Ath-Gouley mines.....	85
11. Curves showing variations in percentage of methane and barometric pressure in the Gabrielle mine, Karwin, Austria.....	87
12. Curves showing relation between length of flame of a blower and variations in atmospheric pressure, November, 1885, to January, 1886, at Hanover mine, Westphalia.....	89
13. Curves showing variations in volumes of methane and carbon dioxide in return air at Tiefbau shaft, Austria, showing relation of variations in atmospheric pressure and also of amount of coal mined.....	91
14. Curves showing variations in percentage of methane and barometric pressure in Du Souich workings, Liévin, France, during April, 1907.....	93
15. Ideal section through coal workings showing area in which flow of gas is affected by removal of coal.....	95
16. Curves showing variations in atmospheric pressure, percentage of methane in return air, and earth tremors at the Herin mine, Anzin, France, August to December, 1886.....	100
17. Curves showing variations in atmospheric pressure, percentage of methane in return air, and earth tremors at Herin mine, Anzin, France, December 6 to 10, 1886.....	101
18. Columnar sections showing stratigraphic position of coal beds in northern anthracite basin.....	106
19. Columnar section of coal measures at Lance mine, Plymouth, Pa....	108
20. Generalized cross section of Lance mine, Plymouth, Pa.....	109
21. Sketch map of worked area in Red Ash bed, Lance mine.....	111
22. Sketch map of worked area in Ross bed, Lance mine.....	112
23. Sketch map of worked area in Bennett bed, Lance mine.....	113
24. Sketch map of worked area in Cooper bed, Lance mine.....	114
25. Sketch map of worked area in Five-foot bed, Lance mine.....	115
26. Sketch map of worked area in Hillman bed, Lance mine.....	117
27. Sketch map showing thickness of valley filling, Lance mine.....	121
28. Map of Red Ash workings in Dorrance mine, Wilkes-Barre, Pa....	130
29. Map of Red Ash workings in Hoyt mine, showing course of air currents sampled in November, 1910.....	135
30. Map of workings in Dunmore bed, Sloan mine.....	140
31. Section through Auchincloss mine near Nanticoke, Pa., showing the great overturn in the lower beds.....	158
32. Tube inserted in coal to ascertain pressure and flow of gas.....	214
33. Relation between depth of mines and amount of methane per ton of coal mined in southern Illinois.....	223

OCURRENCE OF EXPLOSIVE GASES IN COAL MINES.

By N. H. DARTON.

INTRODUCTION.

This report presents the results of an investigation begun by the Government in the summer of 1907, the investigation being started under the immediate supervision of Dr. J. A. Holmes and continued under him as director after it was transferred to the Bureau of Mines in 1910, the field studies being completed in the spring of 1912. The purpose of the investigation was to obtain information on the origin of the inflammable gases in coal and the conditions under which they occur. It was especially intended to ascertain whether there was any relation between the occurrence of gas and the structural or other geologic features of the coal beds. To this end many months were spent in mines and much time was devoted to the examination of mine maps, bore-hole records, and other data made available by the kindness of various coal companies. Two fields of work were selected—one in the northern anthracite basin of Pennsylvania, where the beds are considerably flexed, and the other in the southern part of the bituminous coal field of Illinois, where the beds lie nearly horizontal.

In order to take advantage of results of previous investigations of the same general subject, an extended examination was made of reports of various investigators in Europe and America. A digest of the information obtained from these publications is included in the first part of this report, which constitutes an introduction to the discussion of the conditions governing the occurrence of inflammable gas in coal beds.

THE GASES IN COAL AND THE CONDITIONS CONTROLLING THEIR ESCAPE.

GENERAL CONSIDERATIONS.

All coal contains in its pores and crevices some inflammable gas. This gas is given off when the coal is in the ground, during mining, and for a long time after the coal has been taken from the mine. Peat and marsh muck have the same property; hence the name "marsh gas," which indicates a source in organic material in marshes, is given to methane, the principal inflammable gas in coal.

The amount of gas in coal, the relative proportions of its constituents, and the volume given off vary greatly. In some coals and in some areas the amount of inflammable gas evolved is so small that the mines are classed as "nongaseous." Probably, however, no coal is entirely free from inflammable gas, and if only a small amount now comes from any given coal most of the original content has been lost previously. On the other hand, some coals give off several times their volume of gas that contains a large proportion of methane. In the anthracite region of Pennsylvania there are many mines, from each of which every minute more than 2,000 cubic feet of methane, or nearly 3,000,000 cubic feet every 24 hours, escapes into the air at the mouth of the upcast.

At ordinary temperatures most coals yield an equal volume of gas, or 20 cubic feet to the ton; some yield five times as much, and samples of a few coals have yielded gas at the rate of 150 cubic feet per ton. A volume greater than 100 cubic feet per ton of coal (20 to 25 cubic feet) is, however, relatively rare. The proportion of methane in the gases given off also varies greatly, but from most coals is in excess of 75 per cent.

Inflammable gas is liberated from coal in nearly all mines, but in most shallow mines the volume given off is small, and even in some deep mines of certain districts only moderate ventilation is necessary to keep methane from accumulating sufficiently to be detected with a safety lamp. Unfortunately, however, gas is given off unevenly from place to place in a bed, and in many mines large volumes occur unexpectedly. Wherever inflammable gas is allowed to accumulate an explosion may happen. Formerly all coal-mine explosions were attributed to ignited gas, but now it is known that in large explosions coal dust is the main explosive agent. Most gas explosions are small, but they are frequent in gaseous mines, and many of them are fatal burns. Most afterdamp, or air vitiated by the products of explosion, contains dangerous constituents, the most poisonous of which is carbon monoxide.

CHARACTER OF MINE GASES.

Only a brief outline of the character of the principal gases in mines is given, as the subject is treated in many books.^a Methane (CH_4) is the most important of the mine gases on account of the fact that mixtures of methane and air in certain proportions are explosive. Carbon dioxide (CO_2) is also a common constituent of mine gases even when methane is absent, but only in exceptional instances is the proportion sufficient to be of practical importance. Carbon monoxide (CO) rarely occurs in mines except as a product of incomplete combustion. Oxygen and nitrogen, in the form of air and in proportions other than those in which they occur in air, are contained in gases extracted from coal, and in coal-mining operations large volumes of these gases, as air, are made to pass through the workings to dilute and remove the mine gases. In places mine air carries small amounts of various hydrocarbon compounds besides methane, and hydrogen and hydrogen sulphide occur locally in certain mines.

METHANE.

As expressed by its chemical symbol, CH_4 , methane is a compound of 1 atom of carbon and 4 atoms of hydrogen. It is a colorless, odorless gas, and at ordinary temperatures is chemically inert. On ignition, it unites with oxygen to form carbon dioxide and water vapor. In places it is given off nearly pure by the coal but in most mines it is mixed with more or less carbon dioxide, nitrogen, and oxygen. Its specific gravity in comparison with air is 0.55314; that is, it is slightly more than half as heavy. Seemingly it is not poisonous when mixed with sufficient oxygen to support respiration. Some poisonous effects that have been ascribed to the presence of methane in large quantities are undoubtedly those of suffocation, possibly being increased in certain instances by some poisonous gas. Many miners work all day in air containing 1 to 4 per cent of methane without evidence of injurious effects. An experiment by Haldane^b is on record in which he breathed a mixture of 20 per cent methane and 80 per cent oxygen for 5 minutes without any unfavorable results and a mouse kept in the mixture for 40 minutes was not affected.

CARBON DIOXIDE.

Carbon dioxide (CO_2) is of general occurrence in coal, and is a common constituent of mine air, but the proportion in most mine air is less than 1 per cent. It is without odor or color and is more than half again as heavy as air, its specific gravity being 1.529. It is

^a Much information is given by Beard, J. T., *Mine gases and explosions*, 1908, 402 pp. For a résumé of data relating to gases and methods of analysis see Burrell, G. A., and Siebert, F. M., *The inflammable gases in mine air*: Tech. Paper 39, Bureau of Mines, 1913, 24 pp.; the sampling and examination of mine gases and natural gas; Bull. 48, Bureau of Mines, 1913, 50 pp.

^b Haldane, J. S., *Investigation of the occurrence, properties, and composition of black damp*: Trans. Fed. Inst. Min. Eng., vol. 8, 1894-95, p. 549.

neither explosive nor poisonous but is incapable of supporting life or combustion. In places it accumulates in pits and dug wells. A person descending into such a pit may be suffocated if the proportion of oxygen is much less than 10 per cent.

CARBON MONOXIDE.

Carbon monoxide (CO) is rarely an ingredient of gases issuing from the coal and when it exists in mine air is generally a product of incomplete combustion, whether that of an explosive, of coal or wood in a mine fire, or of explosions of coal dust. It is the principal toxic agent in afterdamp. It is a colorless, odorless gas slightly lighter than air, strongly explosive and extremely poisonous. Air containing a very small proportion of carbon monoxide will cause death especially if the proportion of oxygen is diminished. The amount that can be endured by a person depends mainly on his vitality and the length of time he breathes the gas, but no one can survive more than a few moments in air containing 0.5 per cent. The effect is cumulative and half this proportion would be fatal if breathed for a long time. Removal from the poisoned atmosphere and vigorous long-continued artificial respiration with plenty of air, or better, oxygen, may save a person who has succumbed. The chances of recovery are, however, much less than in suffocation by carbon dioxide or other inert gas.

NITROGEN AND OXYGEN.

Nitrogen is almost invariably a constituent of the gas emanating from coal, in some mines constituting 90 per cent of the gas. In most mines it is mixed with oxygen in various proportions. Air consists of a mixture of approximately 79 per cent nitrogen, 21 per cent oxygen, and 0.03 per cent carbon dioxide, but air in mines loses much of its oxygen by absorption by coal and by various oxidation processes. Air containing less than 7 per cent of oxygen will not support human life.

FIRE DAMP.

The term fire damp is applied to mixtures of methane and atmospheric air; generally to those mixtures that are inflammable or explosive or nearly so. As stated above, pure methane is not explosive because a large volume of oxygen is required for its combustion; neither is a mixture of air and a small percentage of methane explosive. Generally fire damp contains 10 to 30 per cent of methane, for the gas as it emerges from the coal is mixed with more or less air, but in places where the air has been displaced, the proportion of methane may rise much higher; 99.5 per cent was found at a pit in Westphalia in 1894 and 1895. The gas extracted from fresh samples of coal, "blower gas," and gas accumulations in cavities in the roof where little or no

fresh air has had access are much richer in methane than is ordinary fire damp.

The ignition and explosion of fire damp are subjects treated in various books on mine ventilation and mine gases, and their discussion is not within the scope of the present investigation. However, a few general properties may well be restated here. It is generally agreed that a mixture of methane and air containing $9\frac{1}{2}$ per cent methane causes the most violent explosion, and Burrell and Siebert ^a have found that mixtures containing less than 5.5 per cent or more than 12.8 per cent will not explode, the explosive power being greatest when the proportion of methane is between 8 and 12 per cent. The limit of inflammability is reached at about 30 per cent, the mixture burning quietly between this proportion and explosibility.

The limits of explosibility are materially affected by the presence of carbon dioxide ^b and other gases and by coal dust in suspension. The factor of air movement is also of moment. Hydrocarbon gases other than methane increase the explosibility of the mixture, as does also carbon monoxide, but these gases are of infrequent occurrence. Carbon dioxide and nitrogen greatly reduce the explosive range, the most explosive mixture becoming inert when diluted with about 15 per cent of either or both of these gases. A mixture of 1 volume of methane and 9.55 volumes of dry air ignites at 1,200° F., the calculated temperature of the combustion being 3,900° F.^c The pressure developed by such a mixture when it is ignited in a closed space is 102½ pounds to the square inch, which explains why a gas explosion in a mine does so much damage. Probably in a confined space in a mine all the methane does not burn completely in the explosion, for the proportion of oxygen may vary and some of the gas is forced in advance of the explosion wave (and hence is not burned) by the violence of the initial explosion. The presence of coal dust, especially the inflammable dust of bituminous coal,^d is also most important.

NATURE OF GASES IN COAL BEDS.

Many analyses have been made of the gases given off in the mine by coal and by "blowers" and "feeders." Methane is the predominant gas in most places, but its proportion is exceedingly variable, in general ranging from 75 to 99 per cent of the gases as given off. Carbon dioxide is an almost invariable constituent, and the propor-

^a Burrell, G. A., and Siebert, F. M., The inflammable gases in mine air: Tech. Paper 39, Bureau of Mines, 1913, 24 pp.

^b Clement, J. K., The influence of inert gases on inflammable gaseous mixtures: Tech. Paper 43, Bureau of Mines, 1913, 24 pp.

^c Mallard and Le Chatelier, Recherches expérimentales et théoriques sur la combustion des mélanges gazeux explosifs: Annales des mines, vol. 4, 1883, pp. 274-378.

^d Rice, G. S., The explosibility of coal dust: Bull. 20, Bureau of Mines, 1912, 204 pp., 14 pls.; Coal-dust explosions, Miner's Circular 3, Bureau of Mines, 1911, 22 pp.; Frazer, J. C. W., Hoffman, E. J., and Scholl, L. A., Jr., A laboratory study of the inflammability of coal dust: Bull. 50, Bureau of Mines, 1913, 58 pp.

tion, although usually less than 2 per cent, may be much greater in certain places. Nitrogen and oxygen are also present. Carbon monoxide, hydrogen sulphide, and various hydrocarbon compounds are of much less frequent occurrence.

COMPOSITION OF BLOWER GAS.

Chamberlin ^a compared the composition of gas extracted from coal samples in vacuum with gas from "feeders" in mines and bore holes and found that although the feeder gas bore a close resemblance to the gas extracted from lumps of coal and from finely crushed coal during the first few weeks under reduced pressure, it was, however, quite different from gas extracted from finely crushed coal that had lost most of its free gas by exposure to air for a few months. The samples he examined were from the Monongah mine, West Virginia.

The following results of analyses of inflammable gas found in different coal beds in the North of England were made by de la Beche and Playfair in 1846.^b

Results of analyses of inflammable gas from British mines.

Source of sample.	Composition.				
	Methane.	Nitrogen.	Oxygen.	Carbon dioxide.	Hydrogen.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Wallsend colliery, pipe above ground.....	92.8	6.9	0.6	0.3	
Wallsend colliery, Bensham bed.....	77.5	21.1		1.3	
Jarrow colliery, Bensham bed.....	83.1	14.2	.6	2.1	
Hebburn colliery, Bensham bed.....	86.5	11.9		1.6	
Jarrow colliery, five quarter bed.....	93.4	4.9		1.7	
Jarrow colliery, low main bed.....	79.7	12.3	3.0	2.0	3.0
Hebburn colliery, 24 feet below Bensham bed.....	92.7	6.4		.9	
Gateshead colliery, Oakwell gate.....	98.2	1.3		.5	

Le Chatelier from an extensive experience in mines in France and Belgium gives 78 to 99 per cent as the usual proportion of methane in gas from coal and less than 1 per cent as the usual proportion of carbon dioxide.

GAS FROM ENGLISH BLOWERS.

A sample of gas taken in 1893 from a blower at the Hebburn colliery, on the Tyne, England, had the following composition:

Composition of gas from blower, Hebburn colliery.

	Per cent.
Methane.....	78.8
Nitrogen.....	18.6
Oxygen.....	1.7
Carbon dioxide.....	.9

^a Chamberlin, R. T., Notes on explosive mine gases and dusts, with special reference to explosions in the Monongah, Darr, and Naomi coal mines: Bull. 26, Bureau of Mines, 1911, p. 32; reprint of U. S. Geol. Survey Bull. 383.

^b Playfair, Lyon, On the gases evolved during the formation of coal: Mem. Geol. Survey Great Britain, vol. 2, 1846, pp. 460-479; Trans. North of Eng. Inst. Min. and Mech. Eng., vol. 12, 1862, pp. 190-201.

The gas continued to blow for a long time, and analyses made at intervals showed but little variation in the percentage of methane.

Gas from a blower in the Dunraven mine, South Wales, is reported to contain 96.7 per cent methane, 2.8 per cent nitrogen, and 0.47 per cent carbon dioxide.

GAS FROM AUSTRIAN MINES.

The Austrian Fire Damp Commission ^a analyzed many samples of gas from coal mines in Austria; the following is an average of samples from five different sources:

Average composition of blower gas from Austrian coal mines.

	Per cent.
Methane.....	91.0
Carbon dioxide.....	.8
Nitrogen.....	7.1
Oxygen.....	.5
	99.4

A sample from a blower in lignite at Trifail contained 1.4 per cent hydrogen and 7.7 per cent carbon dioxide.

In a mine at Karwin, Austria, a sample of gas from one blower contained 99.10 per cent methane.

Inflammable gas from a dip drift in Glückhilt colliery, Waldenburg, was analyzed by Poleck ^b for the Austrian commission, with results as follows:

Results of analysis of gas from drift in Glückhilt colliery, Waldenburg.

	Per cent.
Methane.....	32.65
Ethane (C ₂ H ₆).....	3.99
Carbon dioxide.....	41.49
Carbon monoxide.....	1.87
Oxygen and nitrogen.....	20.00
	100.00

This is, however, a most unusual combination and the carbon monoxide may be the result of some special reaction.

GAS FROM PRUSSIAN MINES.

In 1882 to 1884 Schöndorff ^c made a series of analyses of gases from coal for the Prussian Fire Damp Commission, with the results following.

^a Atkinson, W. N., Digest of report of Austrian Fire Damp Commission: Trans. Fed. Inst. Min. Eng., vol. 3, 1891-92, p. 531.

^b Redmayne, R. A. S., Modern practice in coal mining, vol. 4, 1911, p. 35.

^c Report of the Prussian Fire Damp Commission (translation), Trans. Fed. Inst. Min. Eng., vol. 3, 1892, pp. 1134-1135.

Results of analyses of samples of inflammable gas from Prussian mines.

Source of sample.	Composition.				
	Methane.	Ethane.	Hydrogen.	Carbon dioxide.	Oxygen and nitrogen.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Blower in Bonifacius mine, Essen.....	90.94		1.40	0.30	7.36
From water in sump, Kreuzgräben mine, Saarbrücken..	^a 57.41		5.68	1.54	35.37
Blower in No. 1 shaft, Consolidation mine, Schalke..	89.88		5.84	.67	3.61
Blower in Shamrock mine, Herne.....	83.97		2.15	.77	13.11
In rise, in Lothringen mine, Castrop.....	^a 27.95		1.35	.45	70.25
In roof of Maria mine, Hönge.....	^a 14.25		.90	.10	84.75
Drift in Zollern mine, Dortmund.....	^a 7.00		.27	.41	92.32
In roof of Tremonia mine, Dortmund.....	^a 9.58		.36	.98	89.08
Drift near blower in Neu Iserlohn mine, Langendreer..	^a 4.75		.09	.13	95.03
Blower in König mine, Saarbrück.....	84.89	1.62		.65	12.84
Blower in Schaumburg mine, Obernkirchen.....	60.46	37.63		2.56	
Blower in mine, Waldenburg, Lower Silesia.....	57.33	.32		.12	42.23

^a Probably more or less diluted by air.

The large proportion of ethane in the blower in the Schaumburg mine at Obernkirchen is a notable feature. The return air from certain levels in this mine was found to carry 0.22, 0.19, 0.35, and 0.10 per cent of ethane, with 0.96, 1.20, 1.02, and 1.11 per cent of methane, and the air had a strong odor of petroleum.

GAS FROM BORE HOLES IN ANTHRACITE REGION, PA.

Some of the holes bored, through the valley filling, into the coal measures near Wilkes-Barre and elsewhere in Luzerne County, Pa., have tapped gas in considerable volume. One notable example is found a few rods southeast of Luzerne, where a pipe in a 98-foot hole sunk 11 feet into coaly shale has been giving off gas for years.

A sample of this gas was collected and analyzed by Chamberlin ^a with the following results:

	<i>Per cent.</i>
Methane.....	91.31
Olefins.....	.49
Carbon dioxide.....	1.59
Carbon monoxide.....	.04
Hydrogen.....	.82
Air.....	1.10
Nitrogen (excess).....	4.65

A sample of gas from a bore hole in coal at the foot of No. 11 slope in the Dorrance mine was collected on September 4, 1908, and analyzed by Chamberlin with the following results:

^a Chamberlin, R. T., Notes on explosive mine gases and dusts, with special reference to explosions in the Monongah, Darr, and Naomi coal mines: Bull. 26, Bureau of Mines, 1911, p. 31.; reprint of U. S. Geol. Survey Bull. 383.

Results of analysis of gas from bore hole in Dorrance mine.

	Per cent.
Methane.....	90.65
Olefins.....	.09
Carbon dioxide.....	.05
Carbon monoxide.....	.00
Hydrogen.....	.77
Air.....	2.73
Nitrogen (excess).....	5.71

GAS FROM FRENCH MINES.

Schloesing ^a has given a series of analyses of samples of strong fire damp from representative French mines as follows:

Composition of samples of fire damp from representative French mines.

Mine.	Methane and other hydro- carbons.	Carbon dioxide.	Oxygen.	Nitrogen.
	Per cent.	Per cent.	Per cent.	Per cent.
Anzin (Herin).....	96.9	0.4	0.5	2.2
Anzin, Renard at 476 m.....	96.4	.0	.1	3.5
Anzin, Renard at 546 m.....	90.2	.3	.0	9.5
Le Grand-Combe, Oules shaft.....	81.8	1.1	9.5	16.6
Le Grand-Combe, Ravin mine.....	86.8	2.6	.4	10.2
Le Grand-Combe, Forêt mine.....	88.8	4.1	.0	7.1
Le Grand-Combe, Pontil shaft.....	88.5	2.7	.1	8.7
Le Grand-Combe, Trets ditch.....	81.1	3.3	.8	14.8
Aciéries de France.....	91.2	1.1	.3	7.4
Campagnac.....	92.5	.7	.0	6.8
Saint Etienne.....	94.6	.4	.1	4.9
Plat-de-Gier.....	78.6	1.0	.2	20.2
Roche la Molière et Firminy.....	89.9	.9	.0	9.2
Rouchamp, Chanols shaft.....	92.7	.7	.0	6.6
Rouchamp, Chanols shaft (duplicate).....	90.6	1.2	.1	8.1
Blanzy.....	55.6	3.7	.9	39.8

The nitrogen is believed to have been derived from outside air from which part or all of the oxygen had been absorbed by the coal, the proportion of argon to nitrogen being nearly the same as in air. The oxygen may also have been derived from outside sources. The methane in some of the samples, notably the last three, had associated with it ethane and possibly other hydrocarbons.

CONDITIONS UNDER WHICH GAS IS GIVEN OFF BY COAL.

Gas escapes from coal mostly by percolating through the minute spaces or pores between the coal grains and also in part through fissures most of which are minute. The pressure of the gas in the coal varies, and the permeability of the solid coal is slight and also variable, two factors that greatly affect the volume of gas given off.

^a Schloesing, T., Sur la composition du grisou: Compt. Rend., vol. 122, 1896, pp. 398-400; Abstract, Trans. Fed. Inst. Min. Eng., vol. 11, 1906, p. 611.

Where coal is shattered or pulverized by pressure and movement gas is freed in large volume as soon as the inclosing solid material is penetrated. Such gas emissions are known as outbursts. Gas accumulates in fissures rapidly if the fissuring is open and so far-reaching as to drain a large surface of coal. Very often, in mining gaseous coal, the gas issues from the fresh working face through many small fissures with a hissing or whispering sound, but some of the outflows are noisy, especially if the gas bubbles through water. Such larger flows of gas are called "blowers" or "feeders." Movements of roof or floor in coal mines and shattering of coal in ribs or pillars increase the general emission of gas. Some of the special conditions under which gas is given off by "blowers," outbursts, and squeezes are described in the following paragraphs.

OCCURRENCE OF "BLOWERS" OR "FEEDERS."

As previously stated, the term "blower" or "feeder" is used by coal miners to denote a flow of gas from a crevice. Usually the flow is from an orifice and the emission is noisy. More blowers occur in or near the roof or floor than in the solid coal, and some are in strata containing no coal. Many blowers are of great volume and of long duration, but most of the stronger ones are short lived and nearly all of them cease in time. A blower at Wellesmeiler, Saarbrücken, has been blowing for more than 50 years. The volume of gas given off by different blowers differs. One persistent blower in the Pelham mine in England in 1847 discharged a volume estimated at 47,000 cubic feet a minute, and one in a mine on the Tyne in England was reported as giving off 6,000 to 7,000 cubic feet per minute. A blower in the Wallsend Colliery, England, is reported by Abel^a as giving off 120 cubic feet of gas a minute. The pressure of some blowers is as high as 50 pounds per square inch.

Blowers or similar noticeable gas emissions are not always an evidence of unusually gaseous coal, for blowers are found in mines where no great volume of methane is emitted by the coal, while, on the other hand, some gaseous mines have no large blowers. In a few mines in France, Belgium, and Germany blowers of carbon dioxide have been found.

Blowers undoubtedly have their origin in crevices in which gas accumulates or in fissures that lead from reservoirs in which gas has accumulated, such as porous sandstone, bodies of crushed coal, or a network of fissures.

A strong blower in the Garwood mine, England, is described by Smethurst.^b It began in a shaft at a point more than 300 feet above

^a Abel, Frederick, Accidents in mines: Proc. Inst. Civil Eng., vol. 90, 1886-87, pp. 160-200; vol. 91, 1887-88, pp. 36-190.

^b Smethurst, W., Final report of Her Majesty's commissioners appointed to inquire into accidents in mines, 1886, p. 146.

the coal bed and had to be piped off as sinking progressed. It burned from the pipes for more than nine years, and at night the light could be seen for 10 miles.

OCCURRENCE OF GAS OUTBURSTS.

CAUSES.

Sudden outbursts of methane in mines indicate somewhat the conditions under which the gas occurs in the coal. These outbursts appear to be due to one of two causes, or a combination of them. One cause of outbursts is a run of coal, which may be produced either by gas or roof pressure, or both, or by mining into a body of shattered coal. Such a run exposes a large area of fresh coal surface, for usually, if the coal is not already much broken, it is greatly shattered by the movement, and thus a large volume of gas is rapidly liberated. The other cause is the opening of a fissure filled with gas under pressure, resulting in a sudden outburst of the gas. The ordinary gas outburst occurs when the pressure of gas exceeds the resistance of the coal, and this condition generally is found when workings in hard coal approach a body of softer or more porous coal containing a large volume of gas under high pressure.

INSTANCES OF SERIOUS OUTBURSTS.

THREE PENNSYLVANIA MINES.

Some very heavy gas outbursts are on record. An outburst at the Otto colliery, 6 miles west of Pottsville, Pa., liberated a body of gas estimated by the mine inspector, Samuel Gray, at 50,000 to 200,000 cubic feet.^a It seemingly came from a great run on a 35° dip in the Mammoth bed and filled an air way for 225 feet. It was carried out by the ventilation without ignition except in one small area where some of it exploded, killing one man and burning several.

A notable outburst in the Knickerbocker mine near Mahanoy City, Pa., in August, 1908, dislodged about 150 cars of coal and gave off a great volume of gas, which suffocated several men. It occurred in a stope of rather steeply dipping beds near the crest of an anticline; the space from which the coal ran was about 6 by 8 feet in cross section and 66 feet long. An outburst in one of the mines near Connellsville, Pa., filled the workings with such a great volume of gas that 100,000 cubic feet of fresh air a minute for three days was required to reduce the proportion of gas in the mine air to a safe limit.

Probably some of the great coal-dust explosions in mines have been started by ignition of a gas outburst. These outbursts occur when the mining has progressed to the vicinity of the gas reservoir, and neither their occurrence nor their ignition can be connected with atmospheric or seismic influences.

^a Gray, Samuel, Explosion at Otto colliery, Pa.: Col. Guard., vol. 8, Nov. 1887, p. 87.

TWO BELGIAN MINES.

At the Agrappe colliery in Belgium, in 1879, an outburst occurred at a depth of 2,000 feet. About 85,000 cubic feet of gas was released so quickly that it reversed the air current and came to the pit mouth, carrying with it a large quantity of fine coal. One hundred and twenty-one men were killed and several others were injured. In the outburst of gas at Bessèges in Belgium, in 1890, 131 men were killed. It is stated that in this outburst 28,000 cubic feet of gas issued rapidly and the overflow continued until in 12 hours 180,000 cubic feet of gas had escaped. From 1850 to 1908 there were 357 serious outbursts in Belgian mines, 32 per cent of them being fatal, and causing the death of 447 men.^a

MORRISSEY, B. C.

An outburst at Morrissey,^b B. C., in 1904, emitted a volume of gas estimated at 5,000,000 cubic feet, and the great mass of fine coal thrown out, amounting to about 3,500 tons, extended 450 feet from the face. Fourteen men were suffocated, but all the lights went out suddenly and there was no explosion.

ABERCARNE MINE, WALES.

A great outburst^c in the Black vein coal at Abercarne mine, south Wales, occurred in a gangway driving through compact coal, when it approached a zone of softer coal. The face blew out and a volume of more than 1,000,000 cubic feet of gas was estimated to have been given off in a short time.

CRUSHED COAL A CAUSE OF GAS OUTBURSTS.

Certain gas outbursts in mines have been found to be closely related to zones in which there has been extensive crushing of the coal, caused by earth movements, especially lateral thrust resulting in folds, buckles, and small faults. In such zones the coal is traversed by numerous crevices which serve as a reservoir for gas given off by the large area of surface exposed by the crushed coal. In an Australian mine the outbursts occurred near such cracks or zones of compression, where the coal was found to be extensively shattered. However, most gas accumulations resulting from such conditions, are found only in deep workings, for, if the beds are near the surface, the gas escapes through the broken strata. The outbursts in the mine at Bessèges in France have been described in detail by Ichon and

^a Described by Arnould, G., *Étude sur les dégagements instantanés de grisou dans les mines de houille du bassin Belge*: Ann. trav. pub. Belgique, vol. 37, 1880, pp. 1-108.

^b Ashworth, James, Outburst of gas and coal at the Morrissey collieries, B. C.: Trans. Inst. Min. Eng., vol. 29, 1904-1905, p. 56.

^c Final report of Her Majesty's commissioners appointed to inquire into accidents in mines, 1886, p. 23.

Lombard.^a These outbursts were numerous and some of them violent, but as most of them gave premonitory indications no fatalities resulted. In one of the outbursts 50 tons of coal was thrown out, and it is estimated that 28,000 cubic feet of methane was given off in the first two and one-half minutes and 175,200 cubic feet in 12 hours. Most of the outbursts were in soft coals, and in coal either normally gaseous or showing less gas than usual, in some instances with harder coal behind. Several occurred in mines where there were small breaks in the coal measures, but most of them were in solid coal. Test holes were not found efficacious either for warning or for decreasing pressure, and they gave a false sense of security.

DISCUSSION OF OUTBURSTS IN BELGIAN COAL FIELDS.

Extended descriptions of the gas outbursts in the Belgian coal field are given by Arnould,^b Dufrane,^c Roberti-Lintermans,^d and Stassart and Lemaire.^e

According to Roberti-Lintermans these sudden outbursts occur in mines along the southern boundary of the field and in only a few beds. Most of the outbursts were at depths of 1,200 to 2,000 feet, but a few were in workings at depths of 2,000 to 3,500 feet. The general dip of the measures has no relation to the accumulations of gas, as the outbursts were as numerous in horizontal as in inclined parts of the same bed. The gas accumulations are, however, closely connected with local disturbances of the strata, such as sudden upturns or faults, folds, or thinnings out of the coal bed, for of 131 outbursts studied only 22 occurred in places where the beds were undisturbed. Most of the outbursts were sudden, and only a few gave premonitory signs. The majority occurred in places where no appreciable volumes of gas had previously been noticed.

OUTBURSTS IN AGRAPPE, PRODUITS, AND MARCINELLE-NORD MINES.

The great outbursts in the Agrappe mine were in horizontal measures where no large quantities of gas had been observed before. An outburst in that mine in October, 1880, threw 1,650 bushels of coal and coal dust into the passages. In June, 1886, in the same vicinity after four test holes had been bored 16 feet without showing signs of

^a Ichon and Lombard, Note sur les dégagements instantanés de grisou aux mines de Bessèges: *Annales des mines*, ser. 9, vol. 1, 1892, pp. 557-603.

^b Arnould, M. G., Étude sur les dégagements instantanés de grisou dans les mines de houille du bassin Belge: *Ann. trav. pub. Belgique*, vol. 37, 1880, pp. 1-100.

^c Dufrane, A., Les dégagements instantanés de grisou: *Bull. Soc. l'indus. min.*, ser. 3, vol. 1, 1887, pp. 507-593.

^d Roberti-Lintermans, Frederic, Les dégagements instantanés de grisou dans les mines de houille de Belgique: *Ann. trav. pub. Belgique*, vol. 52, 1896, pp. 76-244. Review by F. Delafond in *Annales des mines*, ser. 9, vol. 10, pp. 653-669: Translated in *Colliery Guardian*, vol. 74, July 9, 1897, pp. 61-62.

^e Stassart, Simon, and Lemaire, E., Les dégagements instantanés de grisou dans les mines de houille de Belgique: *Ann. des mines de Belgique*, vol. 15, 1910, pp. 93-252, 665-786, 1216-1308, 1737-1810.

extra gas, suddenly 13,750 bushels of coal was thrown out and gas filled the stope for 109 yards. In April, 1885, in a coal mine at Marcinelle-Nord, with several 23-foot bore holes in advance and no signs of extra gas, 3,437 bushels of dusty coal was thrown out and 18 men suffocated. All the gas and coal came from a face only $6\frac{1}{2}$ by $6\frac{1}{2}$ feet. At the Produits colliery, in April, 1891, in a crosscut that had just reached coal, four advance holes through the bed showed so little gas that shots were fired. The shots were followed by a great outburst in which the gas came to the downcast shafts. From one bed in this mine there were 23 small outbursts, at the rate of about two a month. In several instances the methane emanation at the place of outburst notably diminished or ceased for a while before the outburst.

The outbursts of greatest intensity were in crosscuts; they became more frequent and serious as the depths increased, none of importance occurring above a depth of 1,000 feet. In most outbursts the gas is seemingly stored in bodies of permeable coal inclosed in more compact coal; barriers or dams of this character were noted in the Agrappe mine. The high pressure of gas in solid coal noted by various observers indicates that this inclosing material must be almost impervious in order to retain the gas which is under considerable pressure. When the outbursts take place the shattered coal quickly gives off most of its gas. That gas in solid coal can not be drained by bore holes is generally believed, for such holes do not break down enough coal to give outlet to any great volume of gas and, therefore, relieve the pressure only in their immediate vicinity. Gas is given off slowly from compact coal even when the pressure is high, but in approaching zones of shattered coal bore holes in advance of the face might drain some gas. This measure was tried in some of the French and Belgian mines and although in several instances the holes did not even indicate an approaching outburst, in others they delivered a large amount of gas. This question has been discussed by Roberti-Lintermans ^a and by Dufrane ^b in considerable detail. They discuss various methods for avoiding or obviating outbursts.

INVESTIGATIONS OF STASSART AND LEMAIRE.

The detailed investigations of outbursts in Belgian mines by Stassart and Lemaire ^c verified most of the conclusions of previous observers and also brought out various additional facts of interest. They found that the upper series of coals with the largest percentage of volatile combustible matter were the least gaseous and that beds that were very gaseous in one place were much less so in others and below a certain stage the occurrence of gas was without relation to

^a Roberti-Lintermans, Frederic, op. cit., pp. 230-231.

^c Stassart, Simon, and Lemaire, E., loc. cit.

^b Dufrane, A., op. cit., p. 564.

depth. They found also that in some places outbursts were much more likely to occur under the overturned folds and along overthrust faults, especially in overthrust blocks, in all of which the coal was so shattered that gas was given off rapidly. In general, the more gaseous coal was in zones and masses irregularly distributed; this condition is believed to be due largely to areal variations in the amount of gas developed in the original vegetal deposits or kept in them by impervious cover.

OUTBURST DUE TO ROOF FALL OR MOVEMENT OF FACE.

Some outbursts are due to a roof fall, many of which are caused by the pressure of the gas; in others the falling rock may disclose a gas-filled crevice, or a large block of coal may be detached from the face leaving an opening from which gas flows in large volume either from

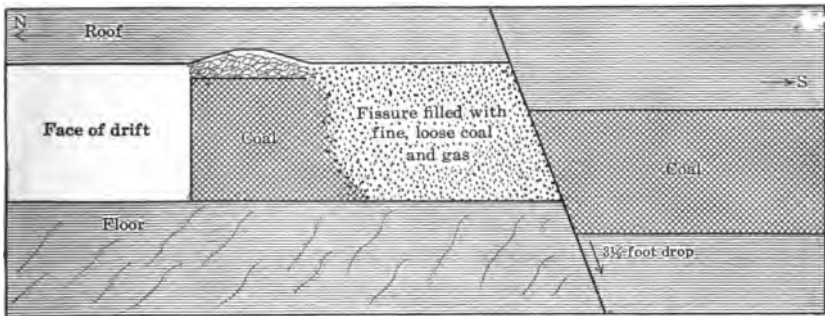


FIGURE 1.—Section showing conditions at place where outburst occurred in the Jarrow mine, England.

crevices or from a mass of shattered coal behind. A good illustration of such a case is given in a description by Buddle^a of an outburst in the Jarrow mine in August, 1830. The outburst came from the face of a drift that was 9 feet wide and 5 feet high. The whole block of coal was forced forward seemingly with great violence, leaving a jagged aperture 9 to 12 inches wide along the roof and down the left side. The block was 4 feet thick, and behind the block a space 7 feet wide extended to a fault with a drop of $3\frac{1}{2}$ feet. This space was filled with disintegrated coal of a sooty appearance. No gas was found before or after the outburst. From this fact it would appear that an isolated reservoir of gas had accumulated in the coal shattered by the fault movement. It was not in communication with a blower, and the gas was quite exhausted by the single eruption. The conditions that existed at this place are shown in figure 1.

^a Buddle, John, Account of an explosion in Jarrow colliery, Aug. 3, 1830: Trans. Nat. Hist. Soc. of Northumberland, vol. 1, 1831, pp. 184-206; Compt. Rend., vol. 2, 1836, pp. 323-324.

GAS FROM SQUEEZES.

The subsidence of an undermined area in a coal mine, known as a squeeze, is attended by the emission of an increased amount of gas, the volume in some instances being large. The emission is caused partly by the crushing of coal in the pillars so that the gas that comes from them gradually under normal conditions is freed quickly. The roof is cracked to some extent and ventilation is usually cut off, at least along some of the main airways.

In the squeeze at the Stevens mine near Pittston, Pa., in July, 1904, a large volume of gas was liberated from workings in the Marcy and Red Ash beds. The air current of 120,000 cubic feet a minute brought so much gas from the squeeze that the return air would flame in a safety lamp set near the fan in the upcast a mile distant. This condition continued for four weeks while the operators were trying to arrest the squeeze.

A squeeze at the Warrior Run mine near Wilkes-Barre, Pa., in August, 1906, was in beds E, F, D, C, and Y, under an area of 40 acres. The affected area was in the north basin which extends from the anticline down to the bottom of the workings. In the crest of the anticline there was 350 feet of rock cover, but the volume of gas was so great that it came to the surface and at one place caused a serious explosion.

METHANE IN COAL.**ORIGIN.**

Methane undoubtedly is a product of the decomposition or molecular rearrangement of organic matter during its transformation into coal, putrefactive bacilli being the cause of the change. It is reasonable to believe that the process may have lasted for some time, but all the evidence now available appears to indicate that it is no longer in progress in the coal. Possibly, however, the diminution of underground pressure in the coal deposits by erosion and mining may be a factor in the disassociation of some as yet unrecognized intermediate compounds with emanation of more or less additional methane.

It appears highly probable that different conditions of bacteriologic action in the original vegetable accumulations have had much to do with the cause of variations in the amount of methane in coal. The first and most important change in the transformation of organic matter into coal is putrefaction, sustained by certain bacilli; this process and others causing changes in the carbon compounds progress until the material reaches the stage of peat. Elaborate studies of these processes made by various observers, notably by Renault ^a in France,

^a Renault, B., Sur quelques microorganismes des combustibles fossiles: Bull. Soc. l'Indus. min., ser. 3, vol. 13, 1899, pp. 965-1161; vol. 14, 1900, pp. 5-159; Les bactériacées des bogheads: Compt. Rend., vol. 124, 1897, pt. 1, pp. 1315-1318.

have aided greatly in determining the organic and chemical conditions involved in the change of vegetable matter to coal. Renault discovered abundant remains of many microorganisms in coal and coaly substances. He and others have found that the putrefaction in vegetable accumulations is caused and sustained by a large variety of organisms, depending somewhat on the nature of the materials which comprise many kinds of plants, on numerous local conditions of temperature, moisture, etc., and on the rate of accumulation. Thus their chemical products may vary considerably.^a The precise chemical reactions are not well known, but in general there is elimination of oxygen and hydrogen with consequent evolution of more or less carbon dioxide and methane. In some stages there are developed by-products that arrest putrefaction, so that it proceeds irregularly in the mass and with varying proportions of resultant gases. The degree of decomposition at the time the coal deposit is buried may be an important factor, and the nature of the covering material, whether sand or clay, and its rate of deposition probably have great influence on the amount of gas retained. Doubtless these two features, the variations in kind and degree of putrefaction and the cover conditions imprisoning the gas in the coal, have most to do with variations in the amount of gas in the coal as now found. Naturally, they varied greatly from place to place and from time to time in basins where vegetable deposits were accumulating and being buried, so that the great variation in volume of methane now found in the coal was largely caused early in the history of the beds. Of course there has also been subsequent escape of gas through porous overlying strata, especially as uplift has brought up the coal measures and they have been more or less bared by erosion. As a result of erosion, the escape of gas has progressed without relation to the original content, and the result gives further complexity to the problem.

RELATIONS TO CLASS OF COAL AND METAMORPHISM.

It does not seem possible to reconcile the methane conditions in coal with generally held theories ascribing the cause of the differences between varieties such as anthracite, bituminous coal, and lignite to fractional distillation after the coal had been buried in the coal measures. It is assumed in such theories that more or less of the volatile matter was baked out by underground heat or "metamorphism" and escaped through porous rocks or joints.^b The baking of coal seen where igneous intrusions have occurred is cited as an illustration of the phenomenon. The nature of the resulting product is supposed

^a See White, David, and Thiessen, Reinhardt, *The origin of coal*: Bull. 38, Bureau of Mines, 1914, 390 pp.

^b Campbell, M. R., *Hypothesis to account for the transformation of vegetable matter into the different grades of coal*: Econ. Geology, vol. 1, 1905, pp. 26-33; White, David, *Some problems of the formation of coal*: Econ. Geology, vol. 3, 1908, pp. 292-318.

to depend upon the stage to which the process advanced, graphite and anthracite being greatly baked, and most lignites having lost little of their original volatile components. Anthracite differs from bituminous coal and lignite in composition mainly in that it contains only about 2 per cent of oxygen instead of 15 to 20 per cent or more. In part of the anthracite basin of Pennsylvania it was found that the amount of methane is far greater than in most bituminous coal fields.

As anthracite in America and Europe contains many volumes of highly volatile methane which must be believed to have been developed during the original formation of the coal, it is impossible to conceive of the distillation of the heavier hydrocarbons of the so-called "volatile matter" of the coal and the retention of the great volume of methane. It implies that vast volumes of "volatile matter" have escaped from coal through strata which are sufficiently impermeable to prevent the escape of methane under high pressure. As shown above, it is most improbable that the methane developed after the metamorphism. Renault, Bertrand, and others have held that although metamorphic agencies may have acted to some extent, the differences in coals are mainly due to difference in character of vegetable accumulations, conditions of deposition, and the nature and degree of biochemical processes that have operated. Doubtless dynamic conditions have caused molecular rearrangement of some of the original components of coal, perhaps with more or less additional methane as byproduct, but the extent and the chemistry of the process have not been elucidated.

CONDITION OF METHANE IN COAL.

The condition under which methane is included in the coal in such large volume is still an unsolved problem. It has been thought that the gas may be dissolved in the coal, as salt dissolves in water and gases intermix, or the gas may be, in part at least, in the form of CH compounds, subject to molecular rearrangement with evolution of methane. The most general opinion is, however, that the gas is inclosed in minute "pores" or crevices of various sizes and manifestly under high pressure, as indicated by the large volume found. It has been suggested that the methane in coal may be in liquid form, but with pressure of 735 pounds to the square inch the temperature of condensation of methane is $-114^{\circ}\text{F}.$ ^a The word "occluded," which has been applied in various senses, is misleading. Chamberlin^b states that even if gases are dissolved in coal

^a Burrell, G. A., and Selbert, F. M., Sampling and examination of mine gases and natural gas: Bull. 42, Bureau of Mines, 1913, p. 107.

^b Chamberlin, R. T., Notes on explosive mine gases and dusts, with special reference to explosions in the Monongah, Darr, and Naomi coal mines: Bull. 26, Bureau of Mines, pp. 16-17; reprint of U. S. Geol. Survey Bull. 383.

they are also held in the minute pores and cavities as well as in the larger crevices, where their presence usually is evident. Analyses of gases given off by samples of coal kept in bottles for a long time show that methane is not forming in the coal, at least under laboratory conditions, for the gas emanation diminishes steadily.

As the volume of gas spontaneously liberated from coal at ordinary temperatures is usually considerably more than the volume of coal and vastly more than that of the interstices or pores in even the most porous coal, its physical condition in the coal is problematical.^a When coal is mined and broken up, the gas escapes rapidly from the pores and crevices, but under ordinary conditions mined coal is in compact lumps and much time is required for the complete liberation of the gas, especially when the tension or gas pressure has diminished.

Chamberlin^b found that during the crushing of the coal in vacuum only about one-quarter as much methane was obtained from the sample as was obtained after it had been bottled for six months in a vacuum. It would seem that crushing of the coal would so shatter all particles that most of the gas could escape at once, but clearly a large amount is held so tightly that much time is required for its liberation.

RELATION OF METHANE TO AGE OF COAL.

There is no relation whatever between the occurrence of methane and the age of the coal, for all coals from Carboniferous to late Tertiary as well as peat and other vegetal accumulations in marshes give off methane. In some districts the younger coals may be more gaseous than the older ones, or vice versa, the difference being due to local conditions.

RELATION OF METHANE TO DEPTH OF COAL.

In general there is more methane in deeply buried coal beds than in beds near the surface, but seemingly the principal factor influencing this condition is the tightness of the cover. There is no evidence that the original content of gas in coal is in any way related to depth, but on the other hand, the proportion varies from bed to bed and in different parts of the same bed without relation to the present position of the surface. Many deep workings are free from large quantities of gas, notably the Monceau-Bayemont, Sacré Madame, Marchienne, and Poirier mines in Belgium. In investigating mines in various parts of the United States the author found that it is not always the deepest bed, or the deepest-lying part of the same bed,

^a This question has been discussed at considerable length by G. Arnould, *Étude sur les dégagements instantanés de grisou dans les mines de houille du bassin Belge.*; Ann. Trav. Pub. Belgique, vol. 37, 1880, pp. 1-108.

^b Chamberlin, R. T., op. cit. p. 32.

that gives off the most methane. Stainier ^a is of the opinion that in wide basins of coal accumulation the chemical changes that produce methane have increased in activity toward the centers of the basins, but these centers are not always at the bottom of the present synclines or structural basins. Moreover, conditions in some of the earliest deposits may not have been such as to produce the most gas. Stainier found that some of the coal areas in Belgium and France are remnants of a wide basin in which the proportion of volatile matter increases toward the center. On the original edges the beds are anthracite, even graphitic, and are thinner. After studying these beds over a wide area, he found that they were much more gaseous toward the center of deposition. Where this part of the field is in the deepest part of the structural basin the more gaseous coal lies at a great depth; but in some districts, on the other hand, the original centers of deposition have been uplifted into anticlines and either lie near the surface on the crests of these uplifts or have been entirely removed by erosion.

RELATION OF METHANE TO STRUCTURAL FEATURES.

Those who have studied the methane problem are generally agreed that structural disturbances alone do not affect the proportion of gas in coal. There are, of course, certain obvious relations when other conditions are equal; the more profound synclines carry the coal so deep that the factor of depth with increased thickness of cover becomes important; workings on anticlines may serve as reservoirs for gas; and zones of crushing contain coal in such condition that it readily gives off a large volume of gas. Many faults serve as channels by which gas may enter workings from lower beds, and shattered measures along faults may carry larger volumes of gas. On the other hand in many places the faults are outlets to the surface which drain off the gas near the plane of movement. A fault may also bring a relatively impervious mass of rock against gaseous beds in such a way as to prevent the escape of gas. In many of the districts where the gas content or pressure is greatest in mines the beds show no signs of structural disturbances or crushing.

In some Prussian mines it was found that in anticlines from which the crests had been removed by erosion, much of the gas had escaped, whereas in the anticlines in which the coal did not reach the surface a large volume of gas remained. This was well shown at Wurm, where coal in the limbs of anticlines that outcrop was free from methane to the deepest workings, whereas in the closed anticlines there was so much gas that it had to be drained off by advance drifts before mining could progress.

^a Stainier, X., Des rapports entre la composition des charbons et leurs conditions de gisement: Ann. des mines de Belgique, vol. 5, 1900, pp. 397-466, 529-590.

In some places more gas is encountered in horizontal beds than in those that are inclined, because the upturned strata are more shattered and the gas drains off through the joints and crevices.

RELATION OF METHANE TO CHARACTER OF COVER.

As no rocks are impermeable to gas, a certain proportion of the original gas content in all coal has come to the surface and drained into the atmosphere, and this process is continually in operation in all coal areas. Time, the porosity and thickness of the overlying deposits, and the volume and pressure of the gas are the important factors in the rate and extent of the drainage. The total amount of gas emanation by this avenue, however, may be only a small proportion of all that is contained in the coal where the bed lies deep and where the cover is fine grained and compact. That tightness of cover is an important feature is shown by the fact that in the Westphalian coal basin 90 per cent of the gas explosions were in the area in which there is a covering of chalk marl which is almost impervious. In the Obernkirchen district the main bed is especially gaseous only where the roof of porous sandstone is covered by Wealden clay, and in those mines in Natal where there is a thick covering of diorite much gas remains in the coal. Doubtless beds containing water impede the escape of gas more than the dry beds, for moisture must greatly diminish the porosity of the strata.

DIFFUSION OF METHANE.

Where methane is given off gradually from the pores and crevices in coal, generally there is perfect diffusion of the gas in the mine air, especially if the ventilation is active. However, where the gas comes from crevices, especially those near the roof, the methane, being lighter than air, rises to the roof and if the air is still remains there in a somewhat pure state. Carbon dioxide, on the other hand, being much heavier than air, tends to accumulate in low places. Of course the tendency of either gas to separate by gravitation is of no import if there is brisk air current, for diffusion is rapid when the air is in motion. Experiments show that three or four hours is required for complete diffusion of a body of methane lying on still air in a mine and that after the gas has been absorbed by the air it can not separate again.^a

The Prussian commission^b made tests at Neunkirchen on the diffusion of methane. The gas was introduced into compartments by a top inlet and in volumes corresponding to 3, 4, and 5 per cent

^a Coquillion, S. S., Sur la diffusion du grisou dans les mines: *Compt. Rend.*, vol. 87, 1878, p. 65.

^b Pellé, M., Expériences sur la diffusion de grisou: *Annales des mines*, ser. 8, vol. 9, 1886, p. 610; *Trans. Fed. Inst. Min. Eng.*, vol. 3, 1892, p. 1141.

of the size of the compartment. The diffusion progressed slowly but was complete in about four hours; in three hours it approached completeness. The composition did not change afterwards.

GASES IN COAL SAMPLES.

The many determinations of the volume and nature of gases in coal samples afford important evidence as to the conditions under which gas is given off in mines and the volume of gas to be expected from coal in mining. Some of these tests have been made by heating the samples and others by collecting gas given off at ordinary temperatures in varying periods of time. The results have shown great diversity in the volume of gas and in its components, not only in coals of the same class but in samples from different places in the same bed. A few representative analyses are given in the following pages. Seemingly no consideration has hitherto been given to the relation between gas content and composition of the coal, a subject that is treated in another section.

GAS FROM SAMPLES OF ENGLISH COALS.

In the earlier determination of gases liberated from coal samples each sample was heated to 212° F., a condition that gives a result not comparable with the emanation of gas at ordinary temperatures in coal mines. The figures obtained by Thomas^a in 1875 from samples of various kinds of English coal are given in the following table:

Gas from samples of English coal.

Kind of coal. ^b	Volume of gas freed per 100 grams.	Approximate relative volume.	Constituent.			
			CH ₄	CO ₂	Oxygen.	Nitrogen.
	c. c.		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Bituminous.....	55.9	0.75	36.42	0.80	0.00	62.78
Do.....	61.2	.82	16.77	2.72	.40	80.11
Do.....	55.1	.74	5.44	1.05	63.76	29.75
Do.....	24.0	.32	22.16	6.09	2.68	69.07
Do.....	39.7	.53	9.43	2.25	31.96	56.34
Semibituminous, steam.....	73.6	.99	12.34	.64	72.51	14.51
Do.....	194.8	2.63	5.04	.33	57.30	7.33
Do.....	250.1	3.38	13.21	.49	81.64	4.66
Do.....	218.4	2.95	5.46	.44	84.22	9.88
Do.....	147.4	1.99	8.90	1.02	67.47	12.61
Do.....	375.4	5.07	9.25	.34	86.92	3.49
Do.....	149.3	2.02	11.35	.56	73.47	14.62
Anthracite.....	555.5	7.50	2.62		93.13	4.25
Do.....	600.6	8.11	4.72		84.18	1.10
Wigan cannel.....	42.13	.57	80.69	6.44		
Do.....	35.06	.47	77.19	9.05		
Scotch cannel.....	16.80	.23		53.94		
Do.....	55.70	.75		84.55		
Whitehill cannel shale.....	55.70	.75		68.75		
Whitby jet.....	30.20	.40	(c)	10.93		

^a Thomas, J. W., Gases inclosed in coals from the South Wales basin, and the gases evolved by blowers and borings into the coal itself: Jour. Chem. Soc., vol. 28, pt. 2, 1875, pp. 793-822.

^b As given in article cited.

^c 86.90 per cent C₂H₄.

In making these tests the coal in one large piece was placed in a vacuum pump; when the air had been exhausted the temperature was raised to 212° F. It was found that the rate of emanation of the gas depended on the hardness of the coal and the quantity of gas it contained.

Bedson and McConnell^a investigated gases contained in coal and coal dust from English mines. Material from a working face in the Ryhope mine gave particularly interesting results. Several clean fragments were heated in a vacuum to 212° F. for 119 hours and the product was tested at intervals. The samples yielded in the end an average of 10 to 11 volumes of gas, of which in the larger amount 1.75 volumes or 16 per cent was methane. The average composition of the gas was as follows:

	Per cent.
Methane.....	16.6
Carbon monoxide	0.1
Carbon dioxide.....	0.7
Oxygen.....	9.4
Nitrogen.....	73.0

Most of the gas was given off in the first 5 hours at 86° to 158° F., but the proportion of methane increased steadily after 5 hours at temperatures of 158° to 197° F., probably because more of the included air had been driven off earlier in the test. However, the proportion of nitrogen increased steadily after 24 hours to the end of the test. The same samples were then powdered and heated in a vacuum for 26 hours at 212° F., a process that liberated less than one-fifth as much additional gas as had been obtained in the first 24 hours in the lump test. This gas contained 17.9 per cent methane, nearly the proportion that was contained in the total volume in the lump test, and nearly three times as much as came off in the last 27 hours in the lump test, showing that the liberation of the methane was greatly furthered by pulverizing the coal. Some heavier hydrocarbons were also liberated from the powdered coal, showing that they are held more tightly. In continuing these investigations^b it was found that different coals showed great diversity as to volume, rate of liberation, and proportions of gaseous constituents.

GAS FROM SAMPLES OF GERMAN COALS.

Many determinations have been made of gas content of German coals. The following series, by Broockmann,^c are of interest as show-

^a Bedson, P. P., and McConnell, W., Notes on the gases inclosed in coal and coal dust: *Trans. Fed. Inst. Min. Eng.*, vol. 3, 1891-92, pp. 307-310.

^b Bedson, P. P., and McConnell, W., A contribution to our knowledge of coal dust: *Trans. Fed. Inst. Min. Eng.*, vol. 7, 1894, pp. 27-530; *Trans. Inst. Min. Eng.*, vol. 24, 1902, pp. 27-40.

^c Broockmann, K., Ueber die in Steinkohlen eingeschlossenen Gasse: *Glückauf*, vol. 35, 1899, pp. 269-274; Abstract in *Trans. Inst. Min. Eng.*, vol. 24, 1902, pp. 18-26.

ing especially low oxygen content. The tests were made of coal fragments freed, as far as possible, from air by standing in vacuum for three days. The gases represented in the table were obtained by heating the coal samples to 212° F. for several days, great precaution being taken to avoid the infiltration of air.

Gas from samples of foreign coals heated to 212° F.

Source of sample.	Approximate relative volume.	Composition of gases evolved.					
		Methane.	Carbon dioxide.	Oxygen.	Nitrogen.	Olefins.	Carbon monoxide.
<i>Westphalia.</i>		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Smithy coal, President pit.....	1.1	96	2		2		
Lean coal, Gabe Gottes bed.....	.6	75	22		3		
Hibernia, coking coal, No. 13 bed	1.2	94	6				
Rhein-Elbe, gas coal, No. 12 bed	.1	12	21		67		
Pluto, coking coal, No. 8 bed....	1.9	87	13				
Cannel coal, No. 7 bed.....	.1	60	40				
<i>Saarbrücken.</i>							
Camphausen, No. 3 bed.....	1.2	84	16				
<i>Upper Silesia.</i>							
Gräfin Laura, Sattel bed.....	.4		60	5	30		5
<i>Obernkirchen.</i>							
Buckeburg, Walder.....	1.1	94	3		3		
Hablschtswald, lignite.....	.6		91				9
<i>England.</i>							
Ryhope, Hutton bed.....	.9	94	3			3	

GAS FROM SAMPLES OF BELGIAN COALS.

Ghysen ^a discusses the results of tests of samples of two coals from Marcinelle-Nord mine, one series being taken from a depth of 3,100 feet in No. 12 shaft and another from the same bed at a depth of 2,920 feet in No. 11 shaft. The tests were made by Fontenille and Lecocq. There had been several strong outbursts of gas from the bed, one of which occurred at a depth of 2,920 feet, blowing out 95 tons of coal. The two shafts are three-fourths of a mile apart and are separated by a great fault; the deeper shaft is generally more gaseous than the other. Samples of about 17½ pounds each, in fragments about 0.6 cubic inch in diameter, were collected every morning, crushed with steel balls for one hour in a pulverizer hermetically tight, and tested twice, at intervals of four hours, in the afternoon. A series of 19 samples from No. 12 shaft yielded gas containing 69.1 to 84.65 per cent methane (mean, 74 per cent), with

^a Ghysen, Henri, Quelques considérations sur les dégagements instantanés de grisou: Rev. univ. de mines, ser. 3, vol. 59, 1902, pp. 1-61.

less than 1 per cent of oxygen and carbon dioxide, the remainder being nitrogen and air. A series of 12 samples from No. 11 shaft yielded gas containing 43.5 to 87.3 per cent methane. The volume of gas in each test was never less than $1\frac{1}{2}$ that of the coal. The increase in quantity of gas given off in the four-hour interval between the two tests was inappreciable. The most important feature is the variation in the volume of gas given off under the same conditions by samples taken at the same hour and from the same place. By crushing a sample in two stages of half an hour each and drawing off the gas after each stage separately a greater total volume of gas was collected than after continuous crushing for an hour. For different samples the composition of the gas differed greatly, a variation that Ghysen believes was due to irregularities in escape of gas through cracks and also to differences in original vegetal composition. Gas was also lost in mining, such loss naturally being uneven. It was found, also, that from samples taken in the same vicinity the volume of gas given off in pulverization varied from two and one-half to five and one-half times that of the coal. Similar tests were made, also, of some samples which had been exposed to air for seven to nine months. In every case they gave off about $1\frac{1}{2}$ volumes of gases, but the composition of these was peculiar in that gas from samples from shaft 11 contained from 9 to 14.75 per cent of methane or other combustible gases; the gas from samples from shaft 12 contained only 1.5 to 2.8 per cent.

Arnould,^a in his study of gas outbursts, found that some coal when placed in a receptacle gave off four times its volume of gas and believed that the amount in the solid coal underground must be greater.

Marsilly^b found that a pressure of 75 pounds to the square inch did not prevent the issue of gas from coal samples, and the Austrian commission found that coal from which gas had been exhausted in a vacuum could not reabsorb as much as it held originally. Coal from the Albert mine at Saarbrücken yielded two to two and one-half volumes of gas.

GAS FROM SAMPLES OF AMERICAN COALS.

Tests made by Chamberlin^c of the volume of gas given off by various coals yielded some interesting results, showing not only the variability in volume and in components, but the variations under

^a Arnould, G., *Étude sur des dégagements instantanés de grisou dans les mines de houille du bassin Belge*: Ann. trav. pub. Belgique, vol. 37, 1880, p. 9.

^b Marsilly, —, *Étude des principales variétés de houilles consommées sur le marché de Paris, [etc.]*: Annales des mines, ser. 3, vol. 12, 1857, p. 360.

^c Chamberlin, R. T., Notes on explosive mine gases and dusts, with special reference to explosions in the Monongah, Darr, and Naomi coal mines: Bull. 26, Bureau of Mines, pp. 17-39; reprint of U. S. Geol. Survey Bull. 383.

different conditions. Under ordinary air pressure the Monongah (W. Va.) coal yielded 0.86 per cent of its volume of methane when powdered to 30-mesh size and only half as much at 10-mesh size. When the coal was placed in a vacuum, the yield was practically the same. Two other samples yielded much less. A sample of anthracite from No. 1 mine, at Nanticoke, Pa., gave off 3.1 volumes of gas in three weeks; during the fourth week a further volume of 0.8, which was nearly pure methane, was given off. The highest proportion of carbon dioxide found was 0.17 per cent of the total volume of gas liberated.

RATE OF ESCAPE OF GAS FROM COAL.

It has been shown that methane and other gases escape continuously from coal while it lies undisturbed underground, while it is being mined, and also for a long time after it has been mined. The rate of escape in general is variable but is without known relation to the mechanical or granular condition of the coal. Coal that had lost much gas in mining continued to evolve methane in the laboratory, one sample liberating three-fourths of its volume in two weeks and one and three-fourths volumes in five months.

Various investigations have shown that there is great variation not only in the total and relative volumes of methane given off by coal samples but in the rate of emanation as well. In general the methane is given off more slowly than the carbon dioxide, oxygen, or nitrogen, so that in the smaller volume of gases extracted in the later part of the test its percentage increased with some samples.

EFFECT OF CRUSHING.

That size of grain is also an important factor has been shown particularly by Chamberlin^a who found the rate of emanation was increased by crushing the coal fine, by the addition of carbon dioxide, and by placing the sample in a vacuum. In his experiments the coal was broken and crushed to coarse powder in an air-tight receptacle connected with a vacuum pump. It is believed that little gas is lost in taking the coal sample from the mine and making the laboratory transfer. The crushing, however, opens a large proportion of pores and minute crevices and thus greatly increases the surface from which the gas is given off. Tests were made with samples under atmospheric pressure as well as in a vacuum. Some samples were pulverized for several hours to ascertain the time factor. A sample from the Monongah mine, West Virginia, was divided into two parts; one part was crushed to 10-mesh and the

^a Chamberlin, R. T. Op. cit., pp. 33-39.

other to 30-mesh size, each part being tested separately. The results^a were as follows:

Relative volumes of methane from 10-mesh and 30-mesh samples of coal.

No. of test.	Conditions of test.	Relative volume of methane to coal.	
		30-mesh sample.	10-mesh sample.
1	In vacuum nearly an hour.....	0.88	0.33
2	In vacuum short time, then in CO ₂ at atmospheric pressure.....	.89	.38
3	In air under atmospheric pressure.....	.86	.41
4	In pure CO ₂ under atmospheric pressure.....	.91	.44
5	In vacuum 3½ hours, then in CO ₂ under atmospheric pressure.....	1.38	.57
6	In vacuum 4 hours, then in hydrogen and CO ₂ under atmospheric pressure.....	1.39	.53

The last two experiments show that the gas does not escape rapidly and that time is an important factor. The experiments as a whole show that if the crushing is not delayed the methane emanation is practically the same whether coal is crushed in vacuum or under normal atmospheric pressure. The gas continues to escape as time passes. The amount immediately liberated also is independent of outside pressure.

Tests were made both in a vacuum and under full atmospheric pressure, and several trials were made to ascertain the time factor up to four hours. Two samples of Monongah coal crushed in vacuum gave the following results:^b

Volumes and analyses of gas from coal crushed in vacuum.

No. of sample.	Relative volume of gas to coal.	Constituent.			
		Methane.	Carbon dioxide.	Air.	Nitrogen (excess).
1.....	0.10	<i>Per cent.</i> 39.65	<i>Per cent.</i> 2.90	<i>Per cent.</i> 48.18	<i>Per cent.</i> 9.27
2.....	.12	40.99	1.56	47.48	10.27

RATE OF ESCAPE FROM LUMP COAL.

In another test, lumps of coal were placed in a receptacle from which the air was exhausted in about 20 minutes, the diminution of pressure amounting to more than 29 inches. The air exhausted in 20 minutes from a vessel containing Monongah coal contained 0.002 volume of methane, whereas coal from the Naomi and the Mansfield mines gave only a trace of methane. A fourth sample from a gaseous

^a Chamberlin, R. T. Op. cit., pp. 21-22.

^b Chamberlin, R. T. Op. cit., pp. 18, 19.

part of the Monongah mine gave off 0.003 volume of methane. A sample of lumps of anthracite coal from No. 1 mine, Nanticoke, Pa., gave off 0.03 volume of methane in 20 minutes and 1.07 volumes in 7 days.

Chamberlin made an extended series of tests^a to ascertain the rate of escape of gas from coal samples during a period of six months. The coal was stored in vacuum bottles and the gas pumped out at intervals for analysis. The volume of gas and rate of escape depended greatly on the size of the fragments. At the end of six months gas was still escaping from lump coal, whereas the liberation of gas from the crushed coal had nearly ceased. Samples of one crushed coal yielded 2.3 to 0.9 volumes of gas, whereas lump samples of the same coal yielded only 0.5 to 0.9 volume of gas. To investigate the volume of gas lost in handling the sample, coal was collected at the face in the mine and placed in cans, which were at once sealed. The gas in the cans was determined a week later, and a month later some of the sample that had been placed in vacuum bottles was tested. Bituminous coal from the Mansfield mine, Carnegie, Pa., gave off 0.55 volume of gas exclusive of nitrogen, and gas escaped more rapidly during the first few days. A sample of gaseous anthracite, however, showed a nearly uniform rate of escape of methane. The rates and percentages of methane emanation from various coals were as follows:

Rate of escape of methane from coal samples crushed to pass a 1-inch mesh and in a vacuum.

Mine.	Kind of coal.	Period of first exposure.	Relative volume of gas to coal (coal=1).	Methane in gas.	Period of second exposure.	Relative volume of gas to coal (coal=1).	Methane in gas.
Monongah.....	Bituminous...	3 weeks...	0.06	<i>Per ct.</i> 35	4 weeks...	0.07	<i>Per ct.</i> 67
Naomi.....	do.....	do.....	.11	55	do.....	.17	85
Mansfield.....	do.....	10 days....	.25	82	10 days....	.11	72
Nanticoke No. 1.....	Anthracite....	1 week.....	1.06	87	1 week.....	.92	94

Mine.	Kind of coal.	Period of third exposure.	Relative volume of gas to coal (coal=1).	Methane in gas.	Period of fourth exposure.	Relative volume of gas to coal (coal=1).	Methane in gas.	Total period of exposure.	Total volume of gas to coal.
Monongah.....	Bituminous...	10 weeks	0.10	<i>Per ct.</i> 76	9 weeks.	0.075	<i>Per ct.</i> 84	6 months	0.30
Naomi.....	do.....	do.....	.25	91	do.....	.18	88	do.....	.71
Mansfield.....	do.....	10 days..	.11	84	do.....	do.....	do.....	30 days..	.47
Nanticoke No. 1.....	Anthracite....	1 week..	.84	94	1 week..	.78	94	do.....	3.60

^a Chamberlin, R. T. Op. cit., pp. 24-31.

Gases other than methane were mainly air, nitrogen, and carbon dioxide. The rate of escape of nitrogen decreased with time, showing that the nitrogen was mainly from residual air that had given up its oxygen to the coal.

Deducting extra nitrogen the proportion of methane rose as high as 98 per cent for some samples; but usually it was between 80 and 95 per cent. The methane percentages were highest in the anthracite sample, which yielded but little carbon dioxide, whereas the bituminous coal from the Mansfield mine contained as much as 13 per cent of carbon dioxide.

Lump coal fresh from some mines continues to evolve gas enough to be lighted in a mine car. Morin ^a cites an instance in which air extracted from the center of a pile of freshly broken coal weighing 3 or 4 tons contained 16 to 26 per cent of methane.

RATE OF ESCAPE FROM PULVERIZED COAL.

In order to compare the rate of escape of gas from finely crushed coal with that from coarse lumps in the test by Chamberlin, just described, samples from other parts of the Monongah mine were crushed to pass through a 10-mesh sieve and then bottled in vacuum. The results were as follows:

Rate of escape of methane from coal crushed to 10-mesh and bottled in a vacuum.^a

Sample.	Period of first exposure.	Relative volume of methane to coal (coal=1).	Proportion of methane in gas.	Period of second exposure.	Relative volume of methane to coal (coal=1).	Proportion of methane in gas.	Period of third exposure.	Relative volume of methane to coal (coal=1).	Proportion of methane in gas.	Total period of exposure.	Total volume of gas to coal.
A.....	6 weeks	0.54	<i>Per ct.</i> 55	10 weeks	0.17	<i>Per ct.</i> 64	10 weeks	0.06	<i>Per ct.</i> 47	6 months	0.77
B.....	do..	.53	63	do..	.08		do..	.07	42		.68

^a Results in round numbers.

In general the proportion of carbon dioxide steadily increased, finally reaching 15 per cent in sample A, and nearly 21 per cent in sample B. Provided that this coal was of the same character as the other, the comparison shows plainly that finely crushed coal gives off much more gas than lump coal; emanation starts more rapidly, and the proportion of methane is larger.

^a Morin, Léon, De l'influence des variations de la pression atmosphérique sur les dégagements de grisou: *Annales des mines*, ser. 10, vol. 16, 1909, pp. 385-487.

A series of tests was made by Chamberlin ^a to determine the rate of escape of methane from coals of various kinds and especially the relative volumes from the same coal in small lumps and in powder. The results are given in figure 2, in which volumes of methane are shown by the vertical scale and the time in weeks by the horizontal scale, the two scales having only an arbitrary relation to each other.

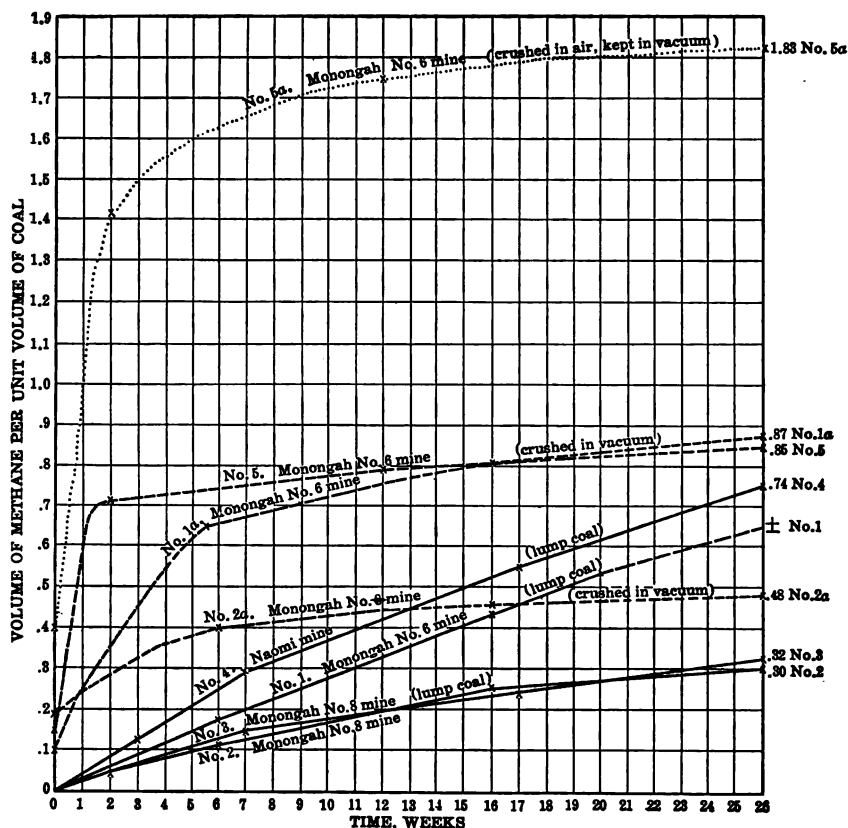


FIGURE 2.—Diagram showing rate of escape of methane from coal under various conditions (after R. T. Chamberlin). — coal size of marbles; --- coal crushed in vacuum; coal crushed in air; X methane determined. Nos. 1 and 1a, same coal; Nos. 2 and 2a, same coal; Nos. 5 and 5a, same coal.

Samples 1 and 1a were taken from the Monongah No. 6 mine. No. 1 was tested in lumps. No. 1a was crushed in a vacuum and then replaced in a vacuum bottle. Samples 2 and 2a were obtained from the Monongah No. 8 mine. The first was in lumps, the sec-

^a Chamberlin, R. T., *Op. cit.*, pp. 83-86.

ond was crushed in a vacuum and placed in a vacuum bottle. No. 3, from the east return airway of the Monongah No. 8 mine, and No. 4, from the most gaseous part of the Naomi mine, were lump samples. Samples 5 and 5a were from the Monongah No. 6 mine. No. 5 was crushed in a vacuum and at once placed in a vacuum bottle, and No. 5a was crushed in air before being placed in a vacuum bottle. The samples were kept in the vacuum bottles for half a year, gas being pumped out at intervals. Samples 1a, 2a, 5, and 5a were crushed to pass through a No. 10 sieve. The volumes of gas liberated in crushing are also included in the volumes shown in figure 2, so that curves 1a, 2a, 5, and 5a start with the quantity liberated.

It is believed that the proportion of gas lost in transferring the powder to vacuum bottles was very small. However, some gas was lost in mining the coal and in breaking it to lumps; also, the gas that accumulated in the cans in five weeks or more escaped, so that the original gas content can only be surmised.

CONCLUSIONS.

The full-line curves in figure 2, representing the methane emanation from lump coal, show that liberation of gas from lump coal even in a vacuum is slow and regular and may continue indefinitely. Sample 1 particularly showed indefinite liberation of methane. Two of the same coals crushed (samples 1a and 2a), as well as other crushed coals (5 and 5a), show a much more rapid rate of emanation at the outset, followed by rapid decline. The methane in crushed samples 1a and 2a was not tested until the sixth week, so that these curves as drawn have too low an initial gradient. This series of tests shows clearly that methane escapes more freely and in larger volume from crushed coal, but that the supply more slowly emitted by the lumps would in time equal that from the powder. Therefore, the extent to which the "pores" of the coal are opened affects the rate of emission, but not the ultimate volume of the gas liberated, so it is not a matter of volume of gas liberated in a given time, but of liberation of all the gas in sufficient time. Chamberlin further summarized the conclusions drawn from these results as follows:

1. Reduction of pressure removes gases from coal slowly—a matter of weeks and months.
2. Finely powdered fresh coal gives off more methane during six months in a vacuum than the same amount of lump coal subjected to the same conditions. But, presumably during a much longer period of time, approximately the same volume of gas should be expected to come off from either type of coal.

3. Reduction of pressure for only a few hours or even a few days is far less effective in extracting the entrapped gas than crushing the coal, either in a vacuum or under barometric pressure.

4. Crushing coal to a certain degree of fineness yields approximately the same volume of methane, whether the operation is performed in a vacuum or in an atmosphere of air or carbon dioxide.

5. Crushing the coal to the degree of fineness of which the crusher is capable removes only a minor proportion of the free gas stored within the coal, for Monongah coal already crushed either in vacuum or under full atmospheric pressure yielded more methane during two weeks in a vacuum than was liberated during the crushing.

The conclusions signify that old coal workings and faces continue to give off gas.

VARIATION IN VOLUME FROM DIFFERENT COALS.

TESTS OF VARIOUS AMERICAN COALS.

Porter and Ovitz ^a have recently tested coals of various kinds from several places in the United States. They supplemented Chamberlin's work, and found that methane and small amounts of carbon dioxide are given off by coal for an extended period under ordinary atmospheric pressure, the amount of gas and its rate of escape being nearly the same as when the coal is kept in a vacuum. They ascertained, also, that oxygen was absorbed rapidly by fresh coal, but in quantities too small to account for the volume of carbon dioxide liberated, if the latter be assumed to result from slow combustion of the coal. They found great variation in the proportion of methane liberated. A Kentucky cannel coal yielded only a trace of methane in nine months, whereas coal from Harrisburg, Ill., gave off one-fourth of its volume in four days, and a coal from Monongah, W. Va., yielded one-third of its volume in the same time. A sample of Pocahontas (Va.) coal yielded little gas, yet mines in the western part of the Pocahontas district contain much gas at times. They found, in general, that at first gas escapes rapidly from coal, then in less and less amounts, with complete cessation in 3 to 18 months.

METHOD OF TESTING.

Porter and Ovitz took about 22 pounds of coal from the clean fresh face, crushed it to pass a $\frac{1}{2}$ -inch screen, and placed it in a crated 5-gallon glass bottle, quickly stoppered with a rubber stopper having a sealed tube in it. This operation occupied 50 to 80 minutes. They estimate that one-fourth of the gas escaped in crushing the coal, as shown by Chamberlin, and one-half to $1\frac{1}{2}$ volumes on continued exposure to air. On arrival at the laboratory some of the air in the bottle was drawn through the tube in the stopper and ana-

^a Porter, H. C., and Ovitz, F. K., The escape of gas from coal: Tech. Paper 2, Bureau of Mines, 1911, 14 pp.

lyzed. The sample was then allowed to stand several months at a temperature of 54° to 88° F. The bottles containing the Pocahontas, Va., and Benton, Ill., coal samples were connected with an oxygen supply and the samples of Sheridan, Wyo., and Harrisburg, Ill., coals with an air supply; the air or oxygen admitted and the gas withdrawn were measured and analyzed. The gas was drawn at intervals from the Pocahontas coal so that at times some pressure may have developed which might have retarded gas emanation. The other samples were tested by withdrawing air from the bottle at intervals. The Harrisburg coal showed gradual diminution of escape

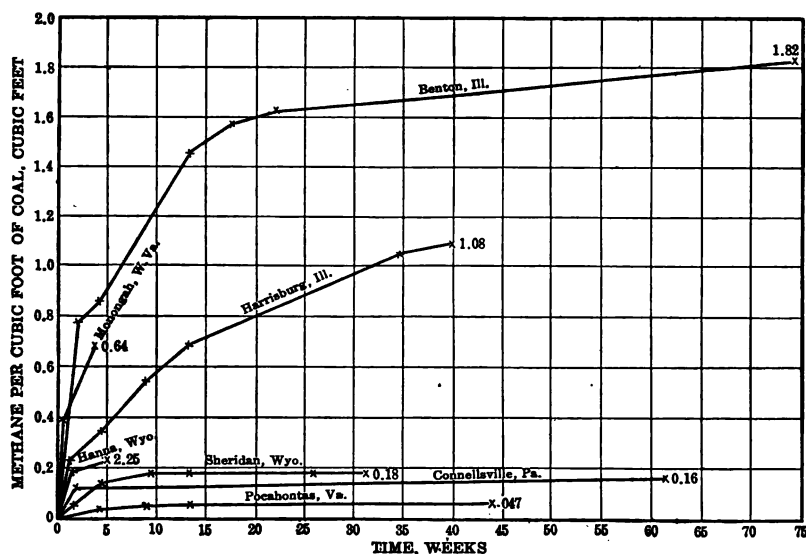


FIGURE 3.—Diagram showing rate of escape and volume of methane from 8 samples of American coals (after Porter and Ovitiz). Time begins 1 hour after mining, hence methane given off in mining is not included.

of methane, long-continued absorption of oxygen, and a steadily increasing liberation of carbon dioxide, results that seemingly indicate that carbon dioxide is held more tenaciously than the other gases. The coals giving the smaller amounts of methane yielded the most carbon dioxide at the start. The coal from Rock Springs and Sheridan, Wyo.; Paintsville, Ky.; Monongah, W. Va.; Harrisburg, Ill.; and Connellsville, Pa., were not tested long but they gave rapid liberation of gas at the start.

RESULTS OF TESTS.

The results of these tests are given in the table following and some of them are shown in figure 3.

Rate of escape of methane and carbon dioxide from coals in small lumps (1 inch).

Source and kind of sample.	Period of time.	Volumes of gas relative to volume of coal (coal=1).		Remarks.
		Methane.	Carbon dioxide.	
Pocahontas, Va., No. 3 bed, Baby mine, semibituminous coal.	First month.....	0.032	0.002	Oxygen admitted as fast as absorbed and gases withdrawn at intervals.
	Second month.....	.009	.000	
	Third month.....	.002	.0002	
	Fourth to ninth month.....	.004	.0016	
	Tenth month.....	.000	.003	
	Total in 10 months.....	.047	.006	
Harrisburg, Ill., No. 5 bed, O'Gara No. 9 mine, bituminous coal.	First 4 days.....	.218	.001	Under flow of air into center and out of top to reservoir.
	Remainder of first month.....	.122	.000	
	Second month.....	.217	.003	
	Third month.....	.125	.000	
	Fourth to eighth month.....	.372	.030	
	Ninth month.....	.029	.019	
	Total in 9 months.....	1.063	.054	
Sheridan, Wyo., No. 9 bed, Dietz No. 2 mine, subbituminous coal.	First week.....	.075	.025	Do.
	Eighth to thirtieth day.....	.078	.008	
	Second month.....	.031	.050	
	Third month.....	.000	.060	
	Fourth to sixth month.....	.000	.158	
	Seventh month.....	.000	.067	
	Total in 7 months.....	.184	.368	
Benton, Ill., Hart-Williams mine, bituminous coal.	First half month.....	.763	.0'0	Connected with oxygen supply at top of bottle and flowing out from below into a reservoir.
	Second half month.....	.113	.000	
	Second and third months.....	.577	.019	
	Fourth month.....	.107	.018	
	Fifth month.....	.065	.007	
	Sixth to seventeenth month.....	.196	.079	
	Total in 17 months.....	1.821	.133	
Rock Springs, Wyo., Rock Spring bed, No. 1 mine, bituminous coal.	First 22 days.....	.015	.075	
	Twenty-third to thirty-first days.....			
	Total in 1 month.....		.075	
Hanna, Wyo., lower bed, No. 2 mine, bituminous coal.	First 10 days.....	.192	.025	
	Eleventh to thirty-first days.....	.033	.002	
	Total in 1 month.....	.225	.027	
Monongah, W. Va., Pittsburgh bed, No. 2 mine, bituminous coal.	First 4 days.....	.390	.000	
	Fifth to thirty-first days.....	.251	.006	
	Total in 1 month.....	.641	.006	
Paintsville, Ky., Flambeau mine, cannel coal.	9 months.....	None.	.035	
Connellsville, Pa., Pittsburgh bed, Leisenring No. 1 mine, bituminous coal.	First day.....			
	Second to eleventh days.....	.122	.006	
	Twelfth to four hundred and twenty-seventh days.....	.038	.007	
	Total in 14 months.....	.160	.013	

TESTS OF ILLINOIS COALS.

An investigation of occluded gases in Illinois coals by Parr and Barker ^a gave results that throw additional light on the volume of gas and the conditions of its emanation.

Samples were collected from the working faces. Each sample was divided into two parts; one was placed in a jar in air, and the other was kept under water in a jar. The samples were allowed to stand for seven months, with the following results:

Volumes of methane given off in seven months by Illinois coals.

Number of sample.	Mine.	Relative volume of methane to coal (coal=1).	
		Dry coal.	Coal under water.
1	Lebanon.....	0.00	0.036
2	Bennett.....	.00	.017
3	O'Fallon.....	.66	.147

The percentages of methane in the gas given off were as follows: No. 1, wet, 56; No. 2, wet, 35.4; No. 3, dry, 11.8; and No. 3, wet, 90.3. Small amounts of carbon dioxide were given off by Nos. 2 and 3, dry.

Tests to determine the effect of oxygen absorption were made with some samples that had been partly air-dried after having been kept sealed 10 months. They were placed in open jars. Two samples were from the Sangamon mine at Springfield, one was from the Chicago & Big Muddy mine near Marion, one from the Davis mine near Duquoin, one from the Suburban mine, near Belleville, one from O'Fallon mine No. 2, one from O'Gara mine No. 8, near Eldorado, and one from the Squirrel Ridge mine near Herrin.

Only the two latter gave off methane, the coal from the No. 8 mine yielding 0.38 volume, or a little more than one-third, and the Squirrel Ridge coal 0.17 volume, or a little less than one-fifth.

As it was realized that considerable methane was lost in collecting the samples of broken fragments at the face of workings, some fresher material was obtained by taking drillings from shot holes. The drillings were run into a flask with suitable tubes, sealed at once, taken to the laboratory with the least possible delay, and the gas removed with a mercury air pump. A second extraction of gas was made several days later. The results follow.

^a Barker, Perry, The occluded gases in Illinois coals: Illinois Geol. Survey Bull. 14, 1908, pp. 204-210; Trans. Am. Inst. Min. Eng., vol. 40, 1909, pp. 24-31; Parr, S. W., and Barker, Perry, The occluded gases in coal: University of Illinois Bull. 32, vol. 6, 1909, 28 pp.

Volumes of methane from drillings of Illinois coals.

Mine.	Town.	Relative volume of methane to coal (coal=1).		Proportion of methane in gas.	
		First test.	Later test.	First test.	Later test.
				<i>Per cent.</i>	<i>Per cent.</i>
Sangamon.....	Springfield.....	0.162	0.081	22.53	59.59
Squirrel Ridge.....	Herrin.....	.252	.254	21.79	86.37
Clifford No. 8.....	do.....	.506	.294	32.44	70.93
Peabody No. 3.....	Marion.....	.021	.001	2.12	1.00
Westville No. 44.....		.262	.255	23.17	19.26

In tests of some laboratory samples of the same coal 2 years old considerable gas but no methane was obtained.

Another test was made of drillings from a 5½-foot hole in the Westville mine, the drillings from the first 2½ feet of the hole being kept separate from those of the last 3 feet. The tests were in two sets; one set was kept in air for 7 days and the other in vacuum for 13 days. The volumes and percentages of methane released were as follows:

Volumes of methane from coal drillings from Westville mine.

Source in drill hole.	7-day test.		13-day test.	
	Relative volume of methane to coal (coal=1).	Proportion of methane in gas.	Relative volume of methane to coal (coal=1).	Proportion of methane in gas.
		<i>Per cent.</i>		<i>Per cent.</i>
First 2½ feet.....	0.149	11.7	0.030	43.7
Last 3 feet.....	.262	23.2	.029	19.3

The percentage of carbon dioxide in the gas was 7.57 and 12.09 in the two 7-day tests, and 33.33 and 32.09 in the 13-day tests.

EFFECT OF OXIDATION.

The early experiments of Von Meyer ^a showed that although coal lost most of its gases, especially methane, when partly exposed, a considerable amount of gas was evolved or could be extracted for years afterwards. His method was to take a sample of nut-sized pieces, place them in a flask, cover them with freshly boiled water, and extract the gases by heating. The following results were obtained from German coals. Samples 1, 2, and 3 were from Zwickau and samples 4 to 9 were from Bochum.

^a Von Meyer, E., Ueber die in Steinkohlen eingeschlossenen Gase: Jour. prakt. C^hemie, vol. 113, 1872, pp. 144-183.

Comparative volumes of gases from fresh and weathered coals.

No. of sample.	Condition of sample.	Relative volume of gas to coal (coal=1).	Composition of gas.				
			CH ₄ .	C ₂ H ₄ and C ₂ H ₆ .	CO ₂ .	O.	N.
			<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
1	Fresh.....	0.285	71.90		2.42	2.51	23.17
	Weathered 5 years.....	.14	3.17	20.08	16.70	4.90	55.15
2	Fresh.....	.40	51.40		.60	.00	48.00
	Weathered 5 years.....	.10	15.88	23.31	7.62	2.44	50.75
3	Fresh.....	.19	45.00		4.02	.62	50.36
	Weathered 1½ years.....	.14	73.16		2.25	.72	23.89
4	Fresh.....	.38	16.65		4.87	2.66	75.82
	Weathered 5 years.....	.32	7.40		11.12	2.88	78.60
5	Fresh.....	.44	31.57		4.82	1.99	60.62
	Weathered 5 years.....	.32	3.31		7.68	2.24	86.77
6	Fresh.....	.32	25.19		2.18	2.12	70.51
	Weathered 5 years.....	.31	6.57		15.84	3.06	74.53
7	Fresh.....	.41	30.25		1.30	1.60	66.85
	Weathered 5 years.....	.29	11.12		4.35	3.35	81.18
8	Fresh.....	.33	5.70		3.72	.39	90.19
	Weathered 1½ years.....	.27	.00		8.49	3.57	87.94
9	Fresh.....	.41	10.65		2.02	.90	86.43
	Weathered 1 year.....	.30	3.43		2.15	3.14	91.28

A series of tests that were made by Parr and Barker ^a to determine the effect of oxidation of coal, showed considerable variation in the amount of methane given off. All three samples were taken from the Squirrel Ridge mine.

Volumes of methane from coal from Squirrel Ridge mine.

No. of test.	Condition of sample.	Conditions of test.	Relative volume of methane to coal (coal=1).	Proportion of methane in gas.
				<i>Per cent.</i>
1	Partly air-dried sample 2 years old.....	Air from sealed container in which sample had been kept for two years.	0.166	2.2
2	do.....	Sealed in air 2 days and air and gas pumped off.	.000	.0
3	Fresh drillings.....	Sealed 14 days and air and gas pumped off.	.252	21.8
4	do.....	Sample in vacuum 12 days after test 3 and gas pumped off.	.253	86.4
5	do.....	Sample sealed in air for 7 days after test 4.	.112	14.1

Tests were also made of samples taken from this and other mines two years before. The samples were quartered, reduced to buckwheat size, and air dried, and although they yielded considerable nitrogen, oxygen, and carbon dioxide, they gave off no methane. The proportion of carbon dioxide varied from 2 to 8 per cent in most of the first tests to 18 to 55 per cent in the later tests, one sample from Springfield yielding as much as 80.5 per cent. Some of the material was from working faces not far from long-standing exposures, the mines having been idle some time.

Screenings from the Westville mine that were 15 and 2 months old yielded no methane in the first test, but two samples of gas removed by vacuum in later tests contained 0.002 and 0.008 volume, respec-

^a Parr, S. W., and Barker, Perry, *The occluded gases in coal*: University of Illinois Bull. 32, vol. 6, 1909, pp. 22-24.

tively. Two-months-old screenings from the Sangamon mine near Springfield, yielded 0.009 volume of methane in the first test and 0.035 volume in a second test. These screenings had been stored outside of the mine. A sample of outcrop coal near Marion, exposed a year, gave 0.006 volume of methane in the first test and 0.001 volume additional in later tests; the proportions of methane in gases given off in the two tests were 0.54 and 1.05 per cent, respectively.

AMOUNT OF METHANE IN MINE AIR.

Much information is on record as to the volume of gas contained in the return air of coal mines. Many tests have been made in mines in Europe, some of them for protracted periods. The results of the author's investigations of numerous mines in the Pennsylvania and Illinois coal fields, which are presented in Parts II and III of this bulletin, show great variation in the total volume of methane emanation from different mines and also from different beds and workings in the same mine. Some parts of mines and some mines that have the most marked manifestations of gas from blowers and local accumulations show smaller total methane emanation than those where the gas is given off generally and continuously.

FACTORS AFFECTING VOLUME.

The proportion of methane in the return air of a mine depends not only on the total volume of gas given off, but also on the volume of air used for ventilation; therefore the most useful figures for comparison are those indicating the volume of methane. Even when the air current is of uniform volume from day to day, the usual condition, the proportion of methane varies, and this variation is sometimes closely related to the amount of coal mined, diminishing in marked degree during a cessation of mining, even for a day. Usually, however, the variation in the volume of methane in the return air is due directly to the variation in the quantity given off by the coal from place to place, increasing as gaseous areas are opened and diminishing as they are drained. Squeezes and outbursts are also important factors, as shown in previous pages. There has been considerable discussion as to what is a gaseous mine, and most classifications proposed have been based on the amount of methane and carbon dioxide in the main returns. This is not a safe basis of comparison, as the danger from gas in a mine depends less on the steady outflow, even if it is large, than on the liability to outbursts and large volume given off in the fresh chambers and headings. Most authorities agree that more than 2 per cent of methane in the return air is an indication of a very gaseous mine.

In most beds of coal there are variations in texture and hardness and in the trend and number of joint planes large and small. The water content in the pores and crevices also varies greatly in different

places. These factors all have important bearing on the variation of the volume of methane contained in coal and the rate of its emanation. The more porous coal may or may not contain a larger volume, but it gives off methane more rapidly. The difference between the total volume of gas given off by the coal and the rate of emanation is usually overlooked in discussing the gas problem. Ordinarily, the coal is regarded as most gaseous that gives the most gas in fresh headings while it is being uncovered, but a more compact coal may contain more gas and give it off gradually and for a long time.

The return air in many mines regarded as nongaseous contains only 0.01 per cent or less of methane, whereas in some gaseous mines main return-air currents of 200,000 cubic feet per minute carry 2 per cent or more of methane. Mention has already been made of mines near Wilkes-Barre, Pa., that discharge 2,000 to 3,000 cubic feet of methane a minute, or nearly 3,000,000 cubic feet in 24 hours; the air in the main upcasts of these mines carries 1 to 2 per cent of methane.

PERCENTAGES IN PRUSSIAN MINES.

Schöndorff ^a has given some interesting figures of percentages of methane in returns in some Prussian mines. In several mines in the Saarbrücken basin, the methane content ranged from 0.05 to 1.46 per cent; in the Aachen basin, from 0.65 to 1.56 per cent; in the lower Rhenish Westphalia basin from 0.002 to 1.428 per cent; and in the lower Silesian basin from 0.27 to 0.92 per cent. He gives the following figures:

Analyses of air in returns in coal mines in Saarbrücken basin, Prussia.

Mine.	Composition of mine air.			
	Methane.	Carbon dioxide.	Oxygen.	Nitrogen.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Albert, east district.....	1.39	0.28	20.19	78.14
Albert, west district.....	.65	.18	20.50	78.67
Van der Heydt, Amselung bed.....	.18	.12	20.40	79.30
Van der Heydt, Beust bed.....	.36	.46	19.62	79.56
Reden, east district.....	.57	.43	20.38	78.62
Reden, west district.....	.82	.43	20.10	78.65
Friedricksthal, small bed.....	.31	.24	20.38	79.07
Friedricksthal, overlying bed.....	.19	.20	20.69	78.92
Dudweiler, shaft 2.....	.70	.25	19.76	79.29
Dudweiler, shaft 1.....	1.08	.09	19.71	79.12
Salzbach, main return.....	.15	.40	20.43	79.02
Altenwald, Nos. 5 and 6 levels.....	.17	.41	20.81	78.61
Altenwald, No. 4 level.....	.30	.87	20.00	78.93
Altenwald, Nos. 1, 2, and 3 levels.....	.12	.46	20.14	79.28
Heinitz, Nov. 10, 1873.....	.13	.53	20.10	79.24
Heinitz, Aug. 18, 1874.....	.18	1.00	19.28	79.54
Heinitz, Sept. 20, 1875.....	.25	.70	19.40	79.65
Heinitz, Sept. 26, 1875.....	.13	.45	19.84	79.58
Dechen, east return.....	.14	.43	19.53	79.90

^a Schöndorff, A., Untersuchung der ausziehenden Wetterströme in den Steinkohlenbergwerken des Saarbrücken: Zeitschr. Berg-Hütt.-Salinenwesen, vol. 24, 1876, pp. 75-125; Chemische Untersuchung von Grubenwettern in Preussischen Steinkohlenbergwerken: Zeitschr. Berg-Hütt.-Salinenwesen, vol. 31, 1883, pp. 435-445; vol. 32, 1884, pp. 509-519.

Some of the results obtained by Schöndorff ^a in testing the methane content in the return air of several of these mines for the Prussian Fire Damp Commission some years later are as follows:

Volumes of methane in return air of different mines.

Mine.	Volume of air delivered per minute.	Proportion of methane in air.	Volume of methane delivered in 24 hours.
	<i>Cubic feet.</i>	<i>Per cent.</i>	<i>Cubic feet.</i>
Albert mine, east district.....	17,000	1.39	340,100
Albert mine, west district.....	15,610	.65	145,700
Van der Heydt mine, Amelung bed.....	34,550	.18	89,060
Van der Heydt mine, Beust bed.....	14,730	.36	76,120
Salzbech mine.....	18,680	.145	38,980
Altenwald mine, Nos. 5 and 6 levels.....	21,920	.165	52,080
Altenwald mine, Nos. 1, 2, and 3 levels.....	37,080	.12	61,980
Altenwald mine, No. 4 level.....	13,080	.30	56,920
Heinits mine.....	27,990	.25	99,760

The total daily volumes of methane from several especially fiery mines were as follows:

Daily volumes of methane from fiery mines.

	Cubic feet per day.
Neu-Iserlohn mine, No. 2 shaft, Lower Rhenish Westphalia.....	897,600
Neu-Iserlohn mine, No. 1 shaft, Lower Rhenish Westphalia.....	305,000
Westphalia mine, Kaiserstuhl shaft, near Dortmund.....	503,700
Albert shaft, Saarbrücken.....	330,000
Gouley-Gemeinschaft-Königsgrube, Aachen.....	295,000±
Friedenshoffnung mine, southern section, lower Silesia.....	256,500
Schaumburger mine, No. 3 shaft, Wealden.....	280,000±

PERCENTAGES IN AUSTRIAN MINES.

The Austrian commission ^b made analyses of the return air in many mines and obtained a great variety of results. One of the most important series reported is from the very gaseous mines of the Karwin-Ostrau district. The results are given in round numbers in the following table:

^a Report of the Prussian Fire Damp Commission (translation); Trans. Fed. Inst. Min. Eng., vol. 4, 1893, p. 639.

^b Atkinson, W. N., Digest of report of the Austrian Fire Damp Commission: Trans. Fed. Inst. Min. Eng., vol. 3, 1891-1892, p. 533; Annales des mines, ser. 9, vol. 1, 1892, p. 343.

Results of analyses of return air in mines in Karwin-Ostrau district, Austria.

Source of sample.	Composition of sample.		
	Methane.	Carbon dioxide.	Oxygen.
<i>Gabrielle mine.</i>			
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Returns from Nos. 12, 25, and 26 beds.....	1.77	0.06	19.61
Returns from No. 23 bed.....	1.25	.14	20.27
Average of the 4 returns.....	1.52	.15	19.87
<i>H. Lariach mine, Karwin East.</i>			
Returns from Nos. 18 and 19 beds.....	1.90	.12	19.54
Returns from No. 8 bed (flat district).....	.83	.18	19.60
Average of 4 returns.....	1.62	.13	19.53
<i>Bettina mine, Dombrau.</i>			
Charles bed, steep dips.....	3.53	.10	19.72
Ignatius bed, steep dips.....	.68	.14	19.64
Average of 6 returns.....	1.72	.10	19.87

PERCENTAGES IN RUSSIAN MINES.

Kotsowsky^a has given the results of a series of analyses of return air in the Donetz basin in Russia. The figures vary greatly as to components, the methane ranging from 0.13 to 2.93 per cent and the carbon dioxide from 0.18 to 1.36 per cent. In the central pit of the Youse mines the methane ranged from 0.32 to 2.50 per cent, and in No. 10 of the Rikowsky mines the air from one gallery carried 2.93 per cent of methane, whereas air at the working face in mine No. 14 contained only 0.13 per cent. The main upcast of the Usine pit carried 0.22 per cent of methane.

PERCENTAGES IN MINES OF SCOTLAND.

Gray^b has given 85 analyses of return air from representative coal and shale mines in Scotland. The depths ranged from 36 to 2,221½ feet, and the air currents had volumes of 1,200 to 46,860 cubic feet a minute, some of them being total returns and others splits. Most of the coal-mine samples contained 0.05 to 0.5 per cent of methane, the content varying without relation to depth, except that some of the very shallow workings were less gaseous than deeper ones. Returns from deep workings in Fifeshire, however, showed no trace of methane. The largest volume of methane was in the main return from the Diamond mine, which carried 185 cubic feet a minute. In general, the shale mines gave off somewhat less methane and carbon dioxide, the volume of methane in the most gaseous return being 93 cubic feet a minute. The carbon-dioxide content was as high as 1 per cent in more than half of the returns.

^a Kotsowsky, N., La composition de l'air des mines dans le bassin du Donetz: Rev. univ. des mines, 3d series, vol. 31, 1895, pp. 50-166. Abstracted in Col. Guard, vol. 70, Nov. 8, 1895, pp. 890-891.

^b Gray, Thomas, Analyses of samples of air from representative mines in Scotland: Trans. Inst. Min. Eng., vol. 39, 1909-1910, pp. 305-312.

RELATION OF VOLUME OF METHANE TO AMOUNT OF COAL MINED OR EXPOSED.**INVESTIGATION IN SAARBRÜCKEN BASIN, GERMANY.**

As the proportion of methane in coal is variable, there is great variation in the amount given off in mining and in preparing coal. An extensive investigation of this branch of the subject was made in the Saarbrücken basin,^a Germany. Twenty-three mines were examined and 652 analyses made. The volume of methane varied from 16½ to 1,060 cubic feet per ton of coal mined, or from one-half to 30 times the volume of the coal, and in one case the volume of gas was 2,160 cubic feet per ton. The methane emanation from the 23 mines varied from 8,870 to 732,566 cubic feet a day, and the total in a year was nearly 2½ billion cubic feet, or 44,000 tons of methane.

EMANATION FROM MINES NEAR WILKES-BARRE, PA.

Several mines in the anthracite region near Wilkes-Barre, Pa., producing about 2,000 tons of coal a day, each give off 2,000 to 3,000 cubic feet of methane per minute or 1,500 to 2,500 cubic feet per ton of coal mined, and the rate of emanation was not diminished greatly after mining had ceased for 30 days. Other mines, however, working the same beds and producing an equal tonnage, give off only 20 to 100 cubic feet a minute or 15 to 75 cubic feet per ton of coal mined. The latter, however, are shallower mines, and the coal may have contained originally as much gas as in the deeper part of the basin.

EMANATION FROM MINES IN SAXONY.

Winkler^b has presented a series of interesting analyses indicating the difference between volume and composition of the ingoing and the outcoming air used in the ventilation of some mines in Saxony. The following is a partial statement of the results:

^a Anon. Quantité de methane (grisou) dégagée par les charbonnages du bassin de Saarbrücken: *Rev. univ. des mines*, ser. 3, vol. 33, 1896, p. 344; *Zeitsch. Berg- u. Hutten*, vol. 43, 1896; *Eng. and Min. Jour.*, vol. 61, 1896, p. 517.

^b Winkler, C. Die chemische Untersuchung der bei verschiedenen Steinkohlengruben Sachsens ausziehenden Wetterströme und ihre Ergebnisse: *Jahrbuch Berghütten in Sachsen*, 1882, pp. 65-84.

Composition and volume of constituents of return air and intake air in coal mines in Saxony.^a

Mine.	Total volume.	Methane.	Carbon dioxide.	Oxygen. ^b	Nitrogen.	Moisture.
	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>
Lugau:						
Upcast.....	8,500	9	41	1,514	6,652	286
Intake.....	7,270		3	1,514	5,696	61
Difference.....	1,230	9	38		956	235
Bockwa-Hohndorf:						
Upcast.....	16,365	25	24	3,046	12,584	684
Intake.....	14,665		6	3,046	11,488	123
Difference.....	1,700	25	18		1,096	561
Deutschland:						
Upcast.....	6,690	9	8	1,209	5,236	226
Intake.....	5,575		2	1,209	4,315	47
Difference.....	1,115	9	6		921	179
Brueckenberg:						
Upcast.....	9,583	25	97	1,721	7,412	328
Intake.....	8,283		3	1,721	6,489	70
Difference.....	1,300	25	94		923	258
Oberhohndorf:						
Upcast.....	6,625	5	23	1,229	5,157	211
Intake.....	5,917		2	1,229	4,637	49
Difference.....	708	5	21		520	162
Van Arnim:						
Upcast.....	19,866	3	205	3,717	15,489	450
Intake.....	17,994		7	3,717	14,018	150
Difference.....	1,972	3	198		1,471	300
Zaukeroda:						
Upcast.....	7,548	1½	33	1,447	5,849	218
Intake.....	6,965		3	1,447	5,457	58
Difference.....	583	1½	30		392	160
Von Burgk:						
Upcast.....	13,882	17	39	2,584	10,816	426
Intake.....	12,149		5	2,584	9,458	102
Difference.....	1,733	17	34		1,358	324
Hainichen:						
Upcast.....	9,537	4	259	1,762	7,727	305
Intake.....	8,479		3	1,762	6,643	71
Difference.....	1,078	4	256		584	234

^a Results are in round numbers per minute.^b It is not known why the amount of oxygen is the same in upcast and intake.

RESULTS OF AUSTRIAN INVESTIGATION.

The Austrian commission^a found that in districts containing much gas the emanation of methane averaged 7,469 cubic feet per ton (21 to 22 cubic feet) of coal mined, whereas in the areas most free from gas the average was 519 cubic feet of methane per ton mined. The effect of the amount of coal mined was evident, but the volume of methane did not vary in direct ratio with it and a variation in the tonnage had to continue for some time to have much effect on the volume of methane. The relation in the Tiefbau mines is shown in figure 13.

^a Report of Austrian Fire Damp Commission: Abstract, Trans. Fed. Inst. Min. Eng., vol. 3, 1892, p. 534.

EMANATION FROM PRUSSIAN MINES.

The Prussian commission^a found that 357 to 2,400 cubic feet of gas was given off per ton of coal mined. Chesneau found 1,377 cubic feet at the Herin mine at Anzin and 883 cubic feet at the Rouchamp mine. In a Westphalian mine it was found that in a bed giving off gas at a moderate rate and without blowers the volume emitted was closely proportionate to the area of freshly exposed coal surface. From his experience in English mines, Thomas was convinced that most of the gas in freshly exposed coal is given off in the first three or four hours.

The following figures have been given by Hilt^b as to the methane given off per ton in mining coal in the Gemeinschaft and the Ath-Gouley mines:

	Cubic feet per ton.
Gemeinschaft, return 2.....	1,060 to 1,152
Ath-Gouley, return 1.....	2,171 to 3,360
Gemeinschaft, lower workings.....	409 to 632
Gemeinschaft, Meister bed.....	2,470 to 2,640

The relation of gas emanation to the area of coal exposed is well illustrated^c in the Gemeinschaft mine in the Grosslangenberg bed which is gaseous throughout. In the anthracite part much of the gas comes from the sandstone roof and a gas emanation may last 10 years, beginning generally with a roof fall. The area is divided by a fault, east of which the bed is "flame" coal, where most of the gas drains off in three years. In 1882 the anthracite workings of this bed gave off 200,000 cubic feet of methane a day. The area of anthracite exposed by workings was 3,300,000 square feet, and for every 1,000 square feet of this surface 60 cubic feet of methane was emitted daily, equal to 219,000 cubic feet in 10 years. If the entire thickness of coal and sandstone be taken as 40 feet, for each cubic foot of material there must have been 6 cubic feet of gas to have sustained this emanation.

In the "flame" coal district, where the daily volume of methane was 130,000 cubic feet, the area of surface exposed in three years of mining was 2,109,000 square feet. The thickness of material holding gas is about $4\frac{1}{2}$ feet and with methane emission of about 60 cubic feet for each 1,000 square feet of area, the total volume of methane from the entire mass of 65,700 cubic feet would be 13.8 cubic feet per cubic foot of coal. This is believed to indicate that in the "flame"

^a Hilt, C., Bericht über Versuche betreffend den Einfluss des wechselnden Luftdruckes auf die Entwicklung des Grubengases: Zeitschr. Berg.-Hütt.-Salinenwesen, vol. 34, 1886, pp. 72-90. Translation by M. Simon, Bull. Soc. l'indus. min., ser. 3, vol. 1, 1887, pp. 595, 625; abstracted in Trans. Fed. Inst. Min. Eng., vol. 4, 1893, pp. 645-646.

^b Hilt, C., Op. cit., pp. 78, 600-615.

^c Report of Prussian Fire Damp Commission (Translation): Trans. Fed. Inst. Min. Eng., vol. 4, 1893, pp. 639-641.

coal district the gas in the coal must have had an average pressure of 207 pounds to the square inch.

In October, 1882, in the Westphalia mine near Dortmund, the workings connected with the Kaiserstuhl shaft, which were in a fiery bed, had an exposed surface of 110,000 square feet.^a It was found that for every cubic foot of solid coal removed 9.7 cubic feet of gas was produced, or 875 cubic feet for every ton mined, and 81 cubic feet of methane for every square foot of coal surface exposed. On the other hand, on January 30, 1884, in the old Westphalia shaft of the same mine, for every ton of coal raised daily only 54 cubic feet of gas was given off (in one district the volume of methane was 60 cubic feet). In one branch airway only recently opened 824 cubic feet of the gas was produced for every ton of coal raised and 236 cubic feet for every square foot of freshly exposed coal.

Some detailed tests made by the same observers in the area mined from the Kaiserstuhl shaft^a showed that gas is not only given off freely by the coal as mined, but continues to be evolved in pillar working. The area investigated was 492 feet long by 256 feet wide, making 125,952 square feet of surface, with a coal content of 20,000 tons. The bed was 6½ feet thick and dipped 65 to 70°. This area was divided by a single-track gangway from which diverged seven roadways 6½ feet wide by 6½ feet high, with pillars 33 feet thick.

The pillars are split by rise drifts and worked retreating. The emanation during the period of observation from December, 1885, to August, 1886, was as follows:

Variations in average emanation of methane in original and pillar workings, Kaiserstuhl shaft.

Kind of mining.	Time of sampling.	Volume of methane—		
		Per ton of coal raised.	Per square foot of surface exposed.	Per square foot of freshly exposed coal.
Whole working.....	1885.			
	December.....	<i>Cubic feet.</i> 657	<i>Cubic feet.</i> 46.9	<i>Cubic feet.</i> 105.7
	1886.			
Passage from whole to working of pillars.....	January.....	880	33.8	110.6
	February.....	833	25.6	110.3
	March.....	876	23.9	109.3
	April.....	1,165	18.0	152.6
	May.....	1,278	14.7	
Pillar working.....	June.....	749		
	July.....	463		
	August.....	438		

^a Report of Prussian Fire Damp Commission (Translation): Trans. Fed. Inst. Min. Eng., vol. 4, 1893, p. 640; Pellé, M., Rapports de la Commission Prussienne du grisou; Étude des variations de la quantité de grisou dégagée par un quartier d'exploitation: Annales des mines, ser. 8, vol. 9, 1886, pp. 615-617.

From December, 1884, to March, 1885, the production averaged 848 tons of coal per month and the area of fresh-coal surface exposed was 6,200 square feet a month. In April, 1885, the production was 571 tons, exposing 4,350 square feet of fresh-coal surface; in May, 446 tons, exposing 1,750 square feet of fresh surface. In all, 38,400 square feet of fresh-coal surface had been exposed. In the three months of 1885 the production varied from 855 to 1,368 tons per month. The total production from December, 1884, to August, 1885, was 8,838 tons, less than half of the total quantity of coal in the area. These figures show conclusively that although the emanation of gas continued during pillar working, it diminished steadily in volume.

The diminution of gas due to Sunday holidays was very noticeable.

EMANATION FROM A MINE AT LIÉVIN, FRANCE.

Morin ^a presents results of determinations of the volumes of methane liberated in part of a mine at Liévin, France, from 1894 to 1908, showing the relation of volume to the area of fresh surfaces of coal exposed. The results were as follows:

Annual variations in volume of methane in return of part of coal mine at Liévin. b

WORKINGS NO. 1.

Year.	Volume of air per minute. c	Proportion of methane in air. c	Volume of methane per minute.	Area of new surface exposed in year.
	<i>Cubic feet.</i>	<i>Per cent.</i>	<i>Cubic feet.</i>	<i>Square yards.</i>
1894.....	89,000	0.19	169	289,472
1895.....	91,532	.27	246	280,026
1896.....	80,600	.31	250	286,638
1897.....	89,000	.36	320	284,612
1898.....	86,680	.45½	395	296,298
1899.....	87,848	.35½	312	275,896
1900.....	119,197	.41	489	279,781
1901.....	135,280	.37½	507	254,378
1902.....	124,355	.38	472	243,654
1903.....	190,360	.25	478	323,554
1904.....	195,665	.20	391	300,033
1905.....	212,935	.18	382	333,511
1906 d.....	179,100	.11½	206	306,256
1907.....	180,263	.15½	210	344,664
1908.....	187,600	.15½	292	339,765

WORKINGS NO. 2 AND NO. 5.

1903.....	88,955	0.07	62	90,669
1904.....	100,785	.11	111	123,572
1905.....	130,380	.19	248	189,459
1906.....	151,180	.16	242	177,981
1907.....	165,985	.26	432	203,732
1908.....	174,245	.30	523	245,730

^a Morin, Léon, *De l'influence des variations de la pression atmosphérique sur les dégagements de grisou*, Annales des mines, ser. 10, vol. 16, 1909, pp. 385-437; abstracted in Eng. and Min. Jour., vol. 90, Sept. 17, 1910, p. 565.

^b Calculated into approximate English measures.

^c Mean for year.

^d Forty-nine days' strike.

In the No. 1 workings, which are the oldest in the mine, the gross volume of methane per day steadily increased from 1894 to 1898 to much more than double, whereas the area of fresh surface exposed increased only $2\frac{1}{2}$ per cent. Then with a diminution of fresh surfaces of about 7 per cent the volume of methane decreased nearly 25 per cent. With slight increase of area in 1900 and great decrease in 1901, the methane volume suddenly increased by more than 50 per cent in 1900 and still more in 1901. Then began an irregular methane decline, notwithstanding a general increase of fresh surface. This diminution is explained by Morin by the fact that workings in adjoining beds became so extensive as to drain some of the gas.

In the No. 5 workings, which are more recent than the No. 1, the methane volume steadily increased with extension of the workings, but at a much more rapid rate. These features and many other observations along various lines led Morin to believe that a large proportion of gas entering the return air is from the roof and floor, and that, other things being equal, the amount of gas increases as the workings are extended and the static equilibrium is disturbed. The figures given above do not therefore show the amount of gas given off from a stated area or tonnage of coal because the sampling was not done near the working faces. The column headed "Area of new surface exposed" therefore shows only the activity of mining.

In his report of investigations of relations between gas emanation and barometric change at Liévin, Morin^a gives some data as to the area of coal surface drained by certain return airways in which the volume of methane was determined. The figures in the following table are calculated from his results, the methane content being taken when it was high at the end of a barometric fall.

Relation of volume of methane to area of coal exposed at Liévin.

Name of return where observation was made.	Methane per 1,000 square feet of coal surface a minute.	Volume of old working.	Name of return where observation was made.	Methane per 1,000 square feet of coal surface a minute.	Volume of old working.
	<i>Cubic feet.</i>	<i>Cubic yards.</i>		<i>Cubic feet.</i>	<i>Cubic yards.</i>
Auguste N.	10.0	1,703	Du Souiche, at 200 meters.	5.2	30,130
Frederic H.	6.4	14,440	Alfred, at 345 meters.	2.4	52,400
Du Souiche, at 476 meters.	14.7	15,720	Alfred, east.	4.0	68,120
Auguste, 31.	2.0	19,650	Alfred, at 476 meters.	2.3	78,000
Edouard couchant. F.	7.5	20,960	East, at 283 meters.	4.3	103,490
Frederic, at 200 meters.	8.2	21,339	South, at 345 meters.	6.4	144,100
Auguste and Du Souiche, 0	5.5	24,628	South, at 282 meters.	5.7	262,000

Some additional data on volume of gas in relation to amount of coal mined in the anthracite coal field of Pennsylvania are given on pages 120 and 143, and in the coal field of southern Illinois, on page 224.

^a Morin, Léon, De l'influence des variations de la pression atmosphérique sur les dégagements de grisou: Annales des mines, ser. 10, vol. 16, 1909, p. 931. Abstracted in Eng. and Min. Jour., vol. 90, 1910, pp. 565-568.

DAILY VARIATIONS IN METHANE EMANATION.

Several extended investigations have been made to ascertain whether gas emanation is proportionate to the variations in activity in mining from day to day. In 1912 the author also obtained data as to the effect of a cessation in Illinois and Pennsylvania due to a strike, with results given on pages 145 and 219.

WINKLER'S INVESTIGATIONS.

Experiments were made by Winkler^a to determine the effect of no mining on Sunday. He analyzed air from mines in three districts in Saxony—Chemnitz, Zwickau, and Dresden—selecting mines that were especially gaseous and others that gave off only moderate amounts. The following results were obtained recalculated to English measures:

Comparison between methane emanation on week days and Sundays in coal mines in Saxony.

Mine.	District.	Volume of methane in mine air.		Volume of upcast.	
		Week day.	Sunday.	Week day.	Sunday.
		<i>Cubic feet a minute.</i>	<i>Cubic feet a minute.</i>	<i>Cubic feet a minute.</i>	<i>Cubic feet a minute.</i>
Lugau.....	Chemnitz.....	9.2	8.3	540	8,980
Bockwa-Hohndorf.....	do.....	25.6	14.2	16,392	12,810
Deutschland.....	do.....	9.3	7.0	6,703	6,110
Brueckenberg.....	Zwickau.....	24.6	11.1	9,600	8,974
Oberhohndorf.....	do.....	4.6	3.7	6,636	6,865
Van Arnim.....	do.....	3.8	5.4	20,848	21,412
Zaukeroda.....	Dresden.....	2.3	1.1	7,560	6,466
Von Burgks.....	do.....	17.4	22.6	13,907	15,455
Hainichen.....	do.....	3.9	6.4	9,572	13,314

TESTS OF PRUSSIAN COMMISSION.

The Prussian commission found in the Heinitz mine in one airway when 727 men were at work that the emanation of methane was 70 cubic feet a minute on Tuesday, whereas on the following Sunday, with no mining, the volume of gas diminished to 40 cubic feet a minute.

TESTS BY HILT.

Tests were made by Hilt^b to ascertain whether the variations in methane during the day were due to barometric change or to mining. For this purpose samples were collected early in the morning before

^a Winkler, C., Die chemische Untersuchung der bei verschiedenen Steinkohlengruben Sachsens anziehenden Wetterströme und ihre Ergebnisse, Jahrb. Berg. Hüttenwesen, Sachsen, 1882, pp. 65-84; abstracted in Eng. and Min. Jour., vol. 34, p. 201.

^b Hilt, C., Bericht über Versuche betreffend den Einfluss des wechselnden Luftdruckes auf die Entwicklung des Grubengases, im Auftrage der Preussischen Schlagwetter Commission, Zeitschr. Berg-Hütt. Salinenwesen, vol. 34, 1886, pp. 72-92. Translation by M. Simon, Bull. Soc. l'indus. min., ser. 3, vol. 1, 1887, pp. 595-625.

work was resumed, at midday, and at 10 o'clock at night. The results are as follows:

Variations in volume of methane given off per minute during different times of the day.

Mine.	Point in mine.	Period of observation.	Average methane emanation.		
			Morning.	Noon.	Night.
		1885.	<i>Cu. meters.</i>	<i>Cu. meters.</i>	<i>Cu. meters.</i>
Gemeinschaft.....	Return 2D.....	Oct. 5 to 10...	184	182	180
Ath-Gouley.....	Return 1D.....	Oct. 11 to 17.	64	62	60
Gemeinschaft.....	430 meters in.....	Oct. 18 to 24.	27	31	31
Do.....	Meister bed.....	Oct. 25 to 31.	29	32½	30

These differences, except in the third set of observations, show no relation to the mining activity, but were found to be closely related to barometric changes.

In the series of methane determinations in the Gemeinschaft and Ath-Gouley mines Hilt ^a gives the following figures, which have been arranged to show the relative amounts of methane given off on the various days of the week for four weeks. They are intended especially to show the effect of the cessation of mining on Sunday:

Methane given off per minute on various days of the week in the Gemeinschaft mine.

RETURN 2D.

Monday.	Tuesday.	Wednesday.	Thursday.	Friday.	Saturday.	Sunday.
<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>
144	163	164	-----	162	149.5	149
162.5	164	164	183	184	171	139
161	155	171	184.5	191	171	186
171	174.5	179.5	161.5	202.5	191	-----
159.5	164	170	176	185	170.5	158

Methane given off per minute on various days of the week in the Ath-Gouley mine.

RETURN 1D.

Monday.	Tuesday.	Wednesday.	Thursday.	Friday.	Saturday.	Sunday.
<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>
64.5	61	55	55	87	51	51.5
54	48.5	45	57.5	63	60.5	54
56	44	45	64	68	68.5	60.5
66	60.5	58	48.5	60	53	-----
60	53.5	51	56	64.5	58	55

In each mine the methane given off increases toward end of week.

The workings in Gemeinschaft mine were largely recent, whereas in the Ath-Gouley mine there are extensive areas of old workings. Some of the variations were due to changes in atmospheric pressure.

^a Hilt, C., Op. cit., pp. 600-615.

TESTS BY LIVING.

Living^a made some tests in part of the Boldon mine to ascertain whether cessation of working for a few days caused diminution in gas emanation. The ventilating air varied in volume from 7,900 to 10,435 cubic feet a minute, and the methane from 0.7 to 0.9 per cent. Some of his figures are as follows:

Variations in volume of methane in Boldon mine during mining and after mining had ceased.

Time of taking test.	Methane.	Extent of operations.
	<i>Cubic feet a minute.</i>	
Wednesday, May 14, 12.40 p. m.	74.7	Mining full capacity.
2.10 p. m.	71.1	Do.
4.10 p. m.	74.7	Do.
5.20 p. m.	74.2	Do.
Thursday, May 15, 11.00 a. m.	72.8	Do.
12.00 noon.	74.1	Do.
1.00 p. m.	72.7	Do.
2.00 p. m.	72.7	Do.
3.00 p. m.	79.6	Do.
4.00 p. m.	80.9	Do.
5.15 p. m.	81.3	Do.
Friday, May 16, 8.00 p. m.	72.1	No mining.
9.00 p. m.	72.8	Do.
10.00 p. m.	72.1	Do.
11.00 p. m.	70.0	Do.
12.00 night.	67.9	Do.
Saturday, May 17, 1.00 a. m.	72.4	Do.
2.00 a. m.	65.6	Do.
3.00 a. m.	67.2	Do.
4.00 a. m.	66.6	Do.
5.00 a. m.	66.9	Do.
6.00 a. m.	66.3	Do.
7.00 p. m.	65.9	Do.
8.00 p. m.	68.2	Do.
9.00 p. m.	66.5	Do.
Sunday, May 18, 6.30 p. m.	67.9	Do.
7.30 p. m.	66.6	Do.
8.30 p. m.	66.0	Do.

^a It is probable that the possible plus or minus due to error of anemometer and of methane determination amounts to fully 2 cubic feet a minute.

An experiment was made on two splits of an air current, one draining a face not worked for nearly a year and the other draining fresh workings, both of about the same area. The first gave off 33 cubic feet of methane a minute, and the fresh one 74 cubic feet, showing that in 12 months the diminution was about half, provided the original gas content was uniform in both splits.

PRESSURE OF GAS IN COAL BEDS.

The gas confined in coal beds must exert a pressure closely proportionate to its relative volume, and as the volume is variable the pressure varies accordingly. Observations made with tubes sunk deep in the coal show pressures ranging from several hundred pounds to the square inch to those almost inappreciable. The causes of these variations are difficult to understand. As the gases are believed to be

^a Final report of Her Majesty's Commissioners appointed to inquire into accidents in mines, 1886, pp. 141-142.

in pores and crevices in the coal, the pressure as mining progresses has a tendency to shatter the coal and throw it out from the face or rib. This condition is common, and in numerous instances men have been killed by masses of coal being suddenly detached. At the Marcinelle-Nord mine, in Belgium, in 1885 about 150 tons of coal was blown out in this way, and numerous large masses were thrown far from the face. Of course at all times the coal in the face, rib, or pillar is under stress from roof pressure and other sources, that has a tendency to break the exposed coal. This pressure is very great in places, especially where the beds dip steeply, some runs of coal being due to roof pressure independent of gas pressure. In such instances also the rapidity with which the gas is given off may be due entirely to the sudden exposure of a large area of coal surface and be no indication of gas pressure.

Observations on pressure of the gas in coal have been made by numerous European observers. During the investigation in southern Illinois the author made a number of tests of pressure in the solid coal, with results as given on page 218.

GAS PRESSURE IN COAL BEDS IN ENGLAND

The first extended investigation of this kind was made by Wood ^a in 1880 in English coal mines. The pressures ranged from 28 to 461 pounds to the square inch. They were measured in tubes tightly tamped in holes 3½ to 47 feet deep that had been bored ahead from working faces into the solid coal. The volume of gas was also determined. The collieries were the Hetton, Eppleton, Boldon, and Harton, all old ones, working bituminous coal 3 to 6 feet thick. The workings were 1,228 to 1,268 feet deep.

METHOD OF CONDUCTING TESTS.

A ½-inch pipe was inserted into the hole to within a few feet of its end. At this inner end was screwed a nut holding a washer nearly the diameter of the bore hole (about 2 inches). The outer end was fitted with a collar, and between the collar and the nut were placed, first, a metal socket, and then several soft-rubber washers that could just be forced into the hole. When the nut was screwed up, a fairly tight fit was effected, but the final sealing was made by filling the space around the socket with Portland cement or oakum in cement. In some cases a wooden plug was also forced into the socket. A gage was attached to the end of the tube for reading pressures, which were observed hourly in most cases. In the deeper holes alternations of rubber washers and wooden plugs were used.

^a Wood, Lindsay, Experiments showing the pressure of gas in solid coal: *Trans. North of England Inst. Min. Eng.*, vol. 30, 1881, pp. 163-256.

RESULTS OBTAINED.

Some of the results obtained were as follows:

Gas pressures observed in English collieries.

Colliery.	Number of hole.	Depth of hole.	Length of hole.	Greatest pressure.	Time required to develop greatest pressure.	Minimum pressure later. ^a	Time required to develop minimum pressure. ^a
		<i>Feet.</i>	<i>Feet.</i>	<i>Pounds.</i>	<i>Hours.</i>	<i>Pounds.</i>	<i>Days.</i>
Hetton.....	1	1,228	9	45	86		
Eppleton.....	1	1,261	34	54½	1		
Do.....	2	1,261	7½	104½	34	90	3½
Do.....	3	1,261	24½	204	106	152	26
Do.....	6	1,261	37	223	174	195	16
Do.....	7		47	235	294	233	18½
Do.....	8		25	221	51	208	8
Boldon.....	1	1,268	19	425	190	425	3
Do.....	2	1,268	32	461	60		
Do.....	4	1,268	23½	381	25	348	6
Do.....	5		26	177	60	175	18
Harton.....	1	1,215	16½	197	135	197	11
Do.....	2	1,215	27½	231	126	230	11
Do.....	3	1,215	37½	296	84	290	7

^a End of experiment.

^b Holes 4 and 5 showed low pressures only 31 and 125 pounds respectively, seemingly due to leakage of gas.

^c Hole 2, 7½ feet deep, showed pressures up to 290 pounds, but was supposed to have leaked into hole 3.

The eight holes in the Eppleton mine were in working faces of four crosscuts far from the shaft and about 150 feet apart, except two that were closer. The coal was in the Hutton bed, which is about 4½ feet thick, and contained 33.3 per cent of volatile matter other than sulphur and moisture. The five holes at the Boldon mine were close together at the end of some workings nearly a half mile from the shaft, holes 1, 2, and 3 being in the same breast at nearly the same height, and Nos. 4 and 5 being about 40 feet distant. They were in the Bensham bed, which is about 5½ feet thick. Holes 1, 2, and 3 in the Harton mine were also near each other, Nos. 1 and 2 being in the same face and No. 3 about 100 feet away. They were in the Bensham bed, 6 feet thick. The pressure was manifested much more rapidly in some holes than in others. From 2 to 16 days was required before the maximum pressure was reached, when the pressure slowly decreased. In some holes oscillation both in increase and in diminution of pressure followed after the maximum. Careful comparisons with barometric readings showed that there was no relation between variations in atmospheric pressure and those shown by the holes.

The highest pressure was obtained at the Boldon mine, where the gage reached 461 pounds in the deepest hole (32 feet), which is about 84 per cent of the pressure that would be due to a column of water the same height as the thickness of the cover. The Boldon mine had been opened only 11 years, whereas the others were 4 or 5 times as old, and this difference may account for the higher pressure. However, there were great differences in pressure in nearby holes and not always in direct relation to their depths, although some of the results

seem to show a fairly close relationship, the ratios of the pressures per square inch being proportional to the square roots of depths.

The direction of hole in relation to cleat seemingly had no effect on the pressures.

The principal features of the more significant test holes are shown in figure 4.

MALLARD'S VIEW OF WOOD'S OBSERVATIONS.

Mallard^a reviewed the results of Wood's observations and deduced from them some hypotheses as to some of the conditions of gas pressure. He compared the gas to water in a porous stratum and believed the same laws of pressure and movement are applicable to it. He showed that under this condition the gas under high pressure back in the solid coal flows toward the face and to the spaces in the test

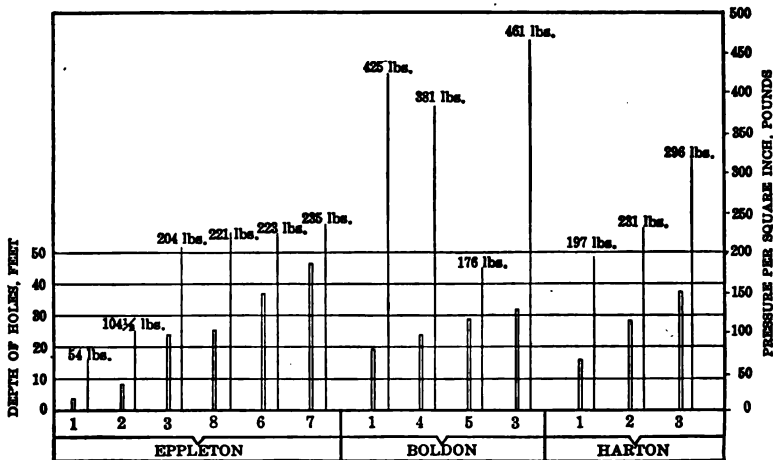


FIGURE 4.—Gas pressures in holes of various depths in English mines. After Wood.

holes and the outward diminution of pressure can be expressed by a mathematical formula. He explained the great variation in pressure from place to place by variations in original gas content, in porosity of coal, and in conditions that have permitted gas to escape from the bed. With gas confined in the coal under favorable conditions for its retention and a thick mass of strata above, the pressures should be expected to rise to 450 to 600 pounds to the square inch or even higher. In general the highest pressures would be found in coal that presents the lowest coefficient of permeability.

However, the pressure observations by Schorn, Watteyne, and Maquet^b do not give results in accordance with the formula of Mallard.

^a Mallard, E., *Expériences sur les pression du grisou dans la houille par M. Lindsay Wood*, *Annales des mines*, ser. 8, vol. 11, 1882, pp. 530-551; Translation, *Trans. North of England Min. and Mech. Eng.*, vol. 33, 1883, pp. 123-132.

^b Schorn, G., Watteyne, V., and Maquet A., *Études sur le grisou: Ann. trav. pub. Belgique*, vol. 44, 1886, pp. 365-367.

A dry bore hole in Stafford Main mine, England, showed a gas pressure of 120 pounds, which was manifested in 20 minutes after the gage was set.

GAS PRESSURE IN COAL BEDS IN WALES.

The British commission on accidents in mines had some interesting pressure tests made in South Wales.^a At the mine of the Harris Navigation Co., the deepest steam-coal workings in South Wales, a 30-foot test hole bored at a depth of 2,133 feet developed a pressure of 150 pounds but found little gas. A second hole bored at a depth of 2,169 feet was 26½ feet deep and showed pressure of 116 pounds. The volume of gas was 0.287 cubic foot an hour. At the Merthyr Vale mine borings 41 feet 2 inches and 49 feet 9 inches at depths of 2,400 and 900 feet showed pressures of 170 and 280 pounds, respectively. At the Celynne mine, near Newport, holes 42 feet, 47 feet 10 inches, and 20 feet 3 inches, at depths of 1,058, 1,480, and 1,500 feet, gave maximum pressures of 129 pounds, 430 pounds, and 318 pounds. The gas volume was at the rate of about 38, 0.4, and 36 cubic feet in 24 hours. A test made in 1886 to show pressure conditions in face of the coal is described in the report of the British commissioners.^b A 4-inch iron tube was tamped into a bore hole 18 inches deep and fitted with a tight piston. A pressure gage was attached to the side of the tube where it entered the coal. It was expected that when the piston was drawn out a partial vacuum would be indicated, but in most tests the gas was given off so rapidly under the reduced pressure that the gage showed no change. When the piston was pushed in again the gas was forced back into the face but escaped from crevices to a distance of 4 to 5 feet about the hole where, on being lighted, it appeared as numerous small flames.

GAS PRESSURE IN COAL BEDS IN BELGIUM.

INVESTIGATIONS OF SCHORN, WATTEYNE, AND MAQUET.

Schorn, Watteyne, and Maquet^c in 1885 to 1887 investigated gas pressures in Belgium in an area where the beds are considerably disturbed. One series of tests was made by Watteyne in shaft 7 in the Belle-Vue mine at a depth of 2,264 feet in the Petite Chevalière, a very gaseous bed. The first holes were made in and near the face of a rock tunnel about 9 feet from the steeply dipping coal bed. The holes passed through all of the coal, at this point about 6 feet thick, and iron tubes were set in the holes through the rock and

^a Final report of Her Majesty's commissioners appointed to inquire into accidents in mines, 1886, pp. 21, 147-148.

^b Final report of Her Majesty's commissioners appointed to inquire into accidents in mines, 1886, p. 21.

^c Schorn, M. G., Watteyne, V., and Maquet, A., *Études sur le grisou: Ann. trav. pub. Belgique*, vol. 44, 1886, pp. 351-451.

tamped in. The holes were drilled at various angles but were grouped in a few square yards, as shown in figure 5.

The pressures are shown in figure 6. A pressure of 555 pounds was obtained in one hole.

In all holes the rise of pressure was very slow at first. After the maximum had been attained in holes 2 and 3 it continued throughout the experiment (10 days), but in No. 1, in which the pressure rose to 330 pounds in 2 days, there was gradual diminution to 150 pounds at the end. In the latter part of the experiment 2 additional holes, Nos. 5 and 6, were bored, and during the boring there was no noticeable diminution of pressure in holes only a few feet distant. The rock tunnel was then driven to the coal, destroying borings Nos. 1, 2, and 3, so that holes 4, 5, 6, 7, and 8, which started obliquely in the floor and roof behind the working face, indicated the pressures (fig. 7). As the coal was uncovered the pressures in these holes diminished gradually, but finally when the coal was completely bared on April 20, they registered 195 and 135 pounds in the two nearest holes (Nos. 5 and 6) and 270 and 412 pounds in those that extended obliquely for some distance to either side (Nos. 4 and 8), the latter not showing any material change from the original pressures. The experiments were repeated when the tunnel approached the Mouton bed 45 feet farther in. Here the pressure went up rapidly the first two days to 525 pounds in one bed and 450 in another and then rose gradually so that at the end of 12 days it was 637 pounds in one hole and 540 pounds in the other.

The results of these observations show (1) that holes did not affect the pressure at nearby holes, except possibly where there were some open cracks between; (2) that uncovering the coal so as to let out a great volume of gas from a large surface had no immediate effect on holes a few yards away; (3) that even the pressures observed, as high as 637 pounds, do not represent the full pressure of the gas dissemi-

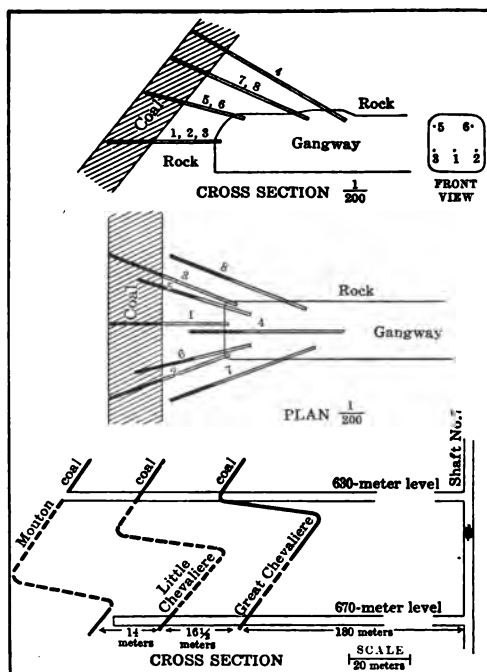


FIGURE 5.—Plan and sections showing location of holes for gas-pressure tests at Belle-Vue mine, Belgium.

nated in the coal if the volume of this gas is four times that of the coal; (4) that the gas is irregularly distributed in coal beds even within short distances; (5) that no law can be applied to this variation in pressure; (6) that the pressure which is established slowly is due to

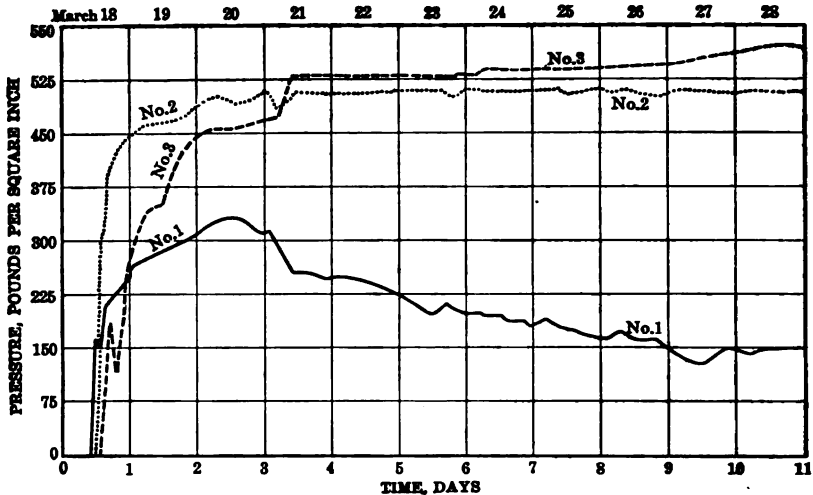


FIGURE 6.—Diagram of pressures in test holes at Belle-Vue mine.

the gas that accumulates little by little in the part of the beds nearest the boring and so establishes an equilibrium against the real pressure of the gas in the bed, the resistance of the compact coal sustaining part of the pressure.

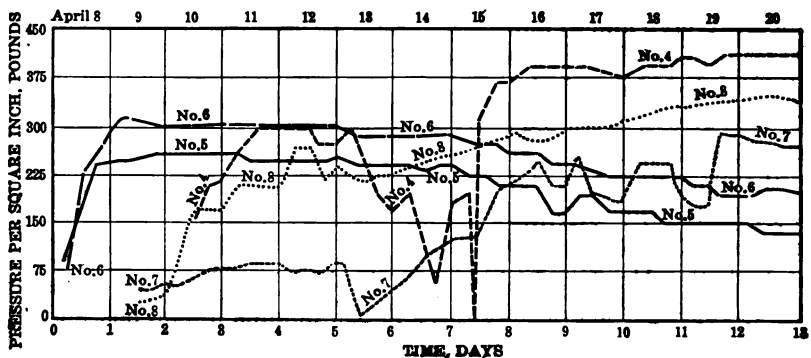


FIGURE 7.—Diagram of pressures in test holes in coal at Belle-Vue mine, Belgium, while the rock tunnel was being continued to the coal.

Ghysen,^a in presenting an abstract of the results of Schorn, Watteyne, and Maquet, notes that the pressures observed at various places do not indicate the danger of gas outbursts, for in the very dangerous

^a Ghysen, Henri, Quelques considérations sur les dégagements instantanés de grisou: Rev. univ. des mines, ser. 3, vol. 59, 1902, p. 40.

bed "Epuisoire," in the l'Agrappe mine, the pressure was less than 15 pounds, whereas in the only slightly gaseous beds at Grand Hornu a pressure of 60 pounds was found and at Marihay a pressure of 225 pounds was observed.

In boring the holes it was noted that those holes showing highest pressure did not give the most gas before being closed, and that most of the gas was liberated in the front part of the hole, possibly because boring cracked the coal in advance and so facilitated the liberation of gas. It was established by these tests that the gas pressure, after attaining a maximum, drops slightly and then remains constant. This may indicate that coal near the hole later absorbs or drains gas from more distant points where there is higher pressure and so establishes an equilibrium. Time is an important factor in the manifestation of the pressure. It was found that when a tube was opened and then reclosed the pressure was attained more rapidly than when the experiment was started. Some fluctuations in the pressure curves appear to indicate waves due to movements of gas through the coal bed.

TESTS BY MAQUET.

Maquet^a made tests of gas pressure at the Beaulieusart mine at Fontaine l'Évêque in the Joseph coal bed in September, 1886. The bed dips 40° and, including partings of carbonaceous shale, is about 5 feet thick. Two holes were bored into the bed at a depth of 1,631 feet at the face of a gangway in the mine, one directed to the northwest, designated as No. 1, and the other to the west, designated as No. 2, No. 2 being south of No. 1. They had the relations shown in figure 8.

Iron tubes were tightly tamped into the holes and were provided with pressure gages which were read at intervals for 11 days. The gage on hole 1 soon indicated a pressure of 187 pounds, and after several days 115 pounds was shown on the gage in hole 2. During the experiment the rise or incline was driven rapidly upward from the

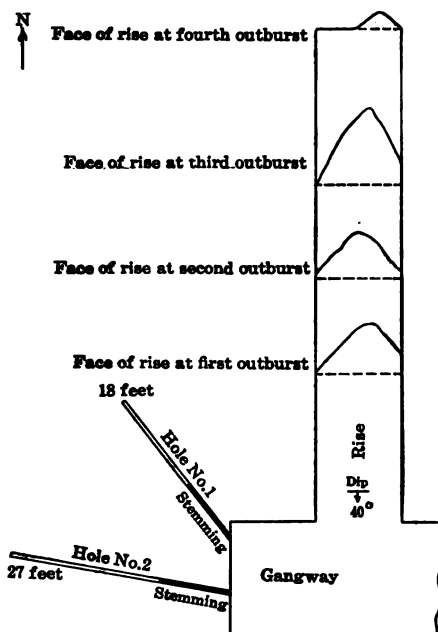


FIGURE 8.—Diagram showing position of test holes at Beaulieusart mine and points at which gas outbursts occurred in driving the rise. After Maquet.

^a Maquet, A., Premier note sur les opérations de sondages et de mesures des pressions du grisou dans Beaulieusart en Fontaine l'Évêque: Ann. trav. pub. Belgique, vol. 44, 1886, pp. 408-445.

gangway and in such manner as to cause gas outbursts of moderate volume at points shown in figure 9. The pressure in the two holes

was affected in a pronounced but varying manner. The lines of pressure are given in figure 9.

It will be noted in these diagrams that the pressure rose rapidly at first, but became much higher in hole 1 than in hole 2. On the second day the pressure (187 pounds) was released in the tube in hole 1 for several minutes without affecting the pressure in hole 2, which was then gradually rising. This experiment tried at other times caused no oscillation in pressure in either tube, and the original pressure was quickly regained.

In the afternoon of the second day a gas outburst occurred, detaching about 1,700 cubic feet of coal, flooding the neighborhood with gas, and stopping observations for 16 hours. Two hours before this outburst the pressure in hole 1 dropped suddenly and continued to fall for several hours after the outburst, till finally on the next morning (September 12) it reached 50 pounds when it ceased falling. In hole 2, so far as could be observed, the outburst had no effect on the steady rise of the pressure, which finally reached 120 pounds, the last few pounds by a rapid rise.

Soon after this there was a second gas outburst much like the first one. The pressure in hole 2 at that instant suddenly dropped from 120 to 44 pounds. The higher pressure was regained soon after the outburst, as shown in figure 9. Meanwhile the pressure in hole 1 showed a slight fall 3 hours before the outburst, then remained stationary and immediately after the outburst rose again. After a short time it became 12 pounds higher

than it had been on the previous day, and then gradually sank and remained stationary at about 34 pounds until near midnight, when it sharply descended to 22 pounds during a third gas outburst in the rise.

This outburst detached about 22,000 cubic feet of coal, but the volume of gas was less than in the previous outbursts. In hole 2 this

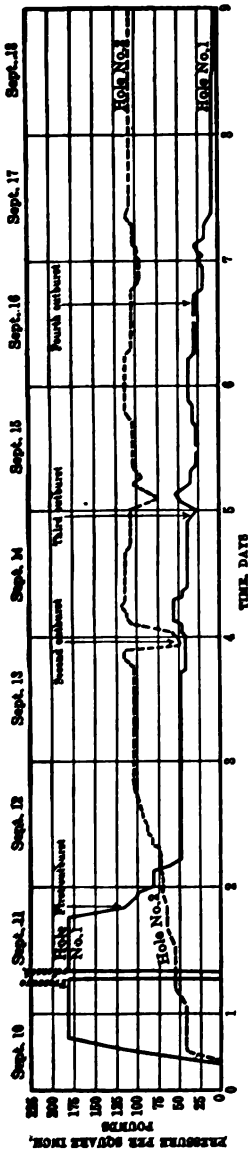


FIGURE 9.—Diagram of pressures in test borings at Beaulieu mine. After Maquet.

outburst was preceded three hours by a slight drop in pressure and quickly followed by a sharp descent of about 30 pounds, then a rise to original pressure and another sharp drop of less amount than the first, after which the pressure finally rose to 115 pounds, where it remained into the next day, September 16. Closely similar oscillations were shown in hole 1 after the outburst, the pressures varying from 50 to 28 pounds. Some of these changes were simultaneous and of similar amounts.

On the afternoon of September 16 there was a fourth small outburst of gas. Three hours after this outburst both tubes showed equal falls of pressure and a series of oscillations. After this the pressure in hole 2 rose a trifle and then remained stationary for several days. The pressure in hole 2, however, rapidly diminished after the oscillations to about 10 pounds, and then diminished very gradually for several days.

CONCLUSIONS FROM RESULTS OF PRESSURE TESTS IN BELGIUM.

From the connection of the oscillations of pressure with the gas outbursts Maquet came to the conclusion that a coal bed with its gas-filled pores acts as an elastic body and transmits wave movements due to pressure variations. He suggests that there are centers of pressure of differing degree and intermediate zones of varying pressure in unstable equilibrium, so that the transmission of waves of varying pressure would not be uniform in speed or amount in all directions. These centers of pressure of varying degree would account for the different pressures found in the holes at different points.

Ghysen^a in discussing the paper does not accept this hypothesis. He holds that the pressure of gas in pores of coal is very high and in stable equilibrium except when cracks give outlet or movement. He explains the rapid drop in pressure in hole 1 as due to a crack opened by mining in the rise and maintains that such cracks started the first outburst. Finally the crack drained off the gas until the pressure dropped to 45 pounds. It did not reach hole 2, but the pressure of the latter was relieved momentarily at the time of the second outburst and following the third and fourth. These breakings of the coal finally led to the declines and ultimate ceasing of pressure in hole 1.

GAS PRESSURE IN COAL IN LIÉVIN MINE, FRANCE.

In 1893 A. Simon,^b chief engineer of the Liévin mine, tested the pressure of gas in solid coal by experiments much like those of Wood and others. The tubes were inserted far out in the workings, at a

^a Ghysen, Henri, Quelques considérations sur les dégagements instantanés de grisou: *Rev. univ. des mines*, ser. 3, vol. 59, 1902, p. 40.

^b Simon, A., Notes sur quelques expériences faites au siège No. 1, des mines de Liévin: *Annales des mines*, ser. 9, vol. 8, 1895, pp. 219-231, abstracted in *Col. Guard.*, Dec. 20, 1895, p. 1164.

depth of about 1,560 feet. The coal contains 30 per cent of volatile matter, and the mine is uniformly gaseous throughout. Two-inch holes were bored into the headings and fitted with copper tubing 0.4 inch in diameter reaching to within $7\frac{1}{4}$ inches of the bottom. The tamping, damp clay, was of different lengths. Pressures were measured with a gage and volumes of gas with a meter. Two series of holes were tried, one in virgin coal, the other in an area where there may have been some fissuring from mining.

TESTS IN FREDERIC BED.

The first series of holes was about 820 feet ahead of the mining in a gangway in the Frederic bed that had been made in the foregoing six months. At that place the bed is about $4\frac{1}{2}$ feet thick, and the test was near a fault where the strata dip 23° . Six holes were drilled into the side of the gangway to depths of $30\frac{1}{2}$ to $39\frac{1}{2}$ feet with 13 feet or more of tamping. Hole 1 was first; 80 feet farther in was No. 5, and then at intervals of 20 to 25 feet Nos. 2, 4, and 3, the latter being at the end of the gangway. It was found that the pressure was not closely related to length of tamping, but the volume of gas for a given pressure was in proportion to the vacant space at the bottom of the hole. The pressures in holes 2, 3, and 5, all with about 13 feet of tamping, were as follows:

Pressures of gas in Frederic bed, Litvin mine.

Number of hole.	Depth of hole.	Area of surface exposed.	Pressure per square inch.	
			March, 1903.	March, 1905.
	<i>Feet.</i>	<i>Square feet.</i>	<i>Pounds.</i>	<i>Pounds.</i>
2.....	$30\frac{1}{2}$	$10\frac{1}{2}$	77	59
3.....	$30\frac{1}{2}$	$10\frac{1}{2}$	59	35
5.....	$39\frac{1}{2}$	$16\frac{1}{2}$	80	$52\frac{1}{2}$

The maximum pressure was attained in four to six days in most cases, but sometimes after the cock had been closed the pressure would rise to the maximum in a few hours. The observations show how slowly the coal gives up its gas, or, in other words, its slight permeability. That the roof is only slightly permeable also was proved by a test hole in which a pressure of only 1.4 pounds per square inch was developed.

TESTS IN ALFRED BED.

The other series of tests was in the Alfred bed at about the same depth as the first. The coal is about 7 feet thick, contains 33 per cent volatile matter, and has a hard floor but a friable roof. The holes were made diagonally into a face at the end of an area of active mining. The results were as follows:

Pressures of gas in Alfred bed.

Number of hole.	Depth of hole.	Depth of tamping.	Area of bottom of hole.	Maximum pressure per square inch developed.	Pressure per square inch after 3 months.
	<i>Feet.</i>	<i>Feet.</i>	<i>Feet.</i>	<i>Pounds.</i>	<i>Pounds.</i>
1.....	23	21½		64½	49
2.....	30	27		85½	59
3.....	44	42	1½	96½	59

It is to be noted that the difference in maximum pressure in holes 2 and 3 is small, notwithstanding the great difference in coal surface exposed. This relation is variable. The rapid diminution of pressure in three months was due to the greater permeability of the bed and to some fissuring from the advancing workings. The gas in this series in hole 3 showed higher pressure than in the holes in the Frederic bed and afforded over 0.35 cubic foot a minute, or nearly fifty times as much in relative volume when the small area of coal exposed (1½ square feet) is considered. The maximum pressure observed in the mine was 105 pounds per square inch in a hole 39 feet deep and is very much less than those recorded by L. Wood in his tests in an English mine and by other observers in French and Belgian mines. It was found that the variation in pressure was not uniform, and although it increased with depth of hole in most cases, the results do not accord with Mallard's^a formula. The long gangway in the Frederic bed where the six test holes were made drained the gas very slowly for about two years, and as shown by the table on page 70, the pressure decreased only about one-third in that time. Mallard's view that the volume of gas increased with pressure was not sustained by Simon's tests, which showed that with practically the same pressures the gas volume per square foot of coal exposed was fifty times as much in the Alfred bed, which is disturbed, as in the Frederic bed, which is undisturbed.

TESTS IN LEONARD BED.

In 1907 Morin^b made some determinations of pressure of gas in coal in a gangway down the dip in virgin coal of the Leonard bed, where the volume of gas in the return air current was 31 cubic feet per minute. Into this bed bore holes were driven to various depths. A copper tube was inserted into each bore hole to within 8 inches of the bottom. The space about the tube was then carefully tamped with clay to within 4 feet of the bottom.

In one of these holes, which was 25 feet deep, the pressure of the gas was 10.6 pounds per square inch, diminishing to 7 pounds two weeks

^a Mallard, E., *Expériences sur la pression du grisou dans la houille* par M. Lindsey Wood: *Annales des mines*, ser. 8, vol. 1, 1882, pp. 530-551.

^b Morin, Léon, *op. cit.*, pp. 385-437.

later. The volume of gas given off at 7 pounds pressure was $23\frac{1}{2}$ cubic feet per minute from an exposed surface of $2\frac{1}{2}$ square feet. After allowing this gas to flow freely for two hours, the pressure gage was again attached and in 45 minutes the gas regained its original pressure. The pressure remained at this point for three weeks, but a week later it diminished to 4 pounds and still later to less.

Compared with the results obtained by Simon in 1893, the pressures are lower, the flow of gas more copious, and the renewal of pressure more rapid. Further tests of the same holes indicated a failing pressure, but without corresponding decrease in volume, which appears to prove that when gas drainage begins the loss is slow at first and then rapidly increases as the drainage becomes well established.

GAS PRESSURES IN COAL AT ST. ÉTIENNE.

In investigating the cause of several coal and gas outbursts at St. Étienne, France, Petit^a made some tests of pressure of gas in the coal similar to those made by Wood, Schorn, Maquet, Simon, and others. The bed tested was No. 13, which averages about 14 feet thick and is 380 feet below sea level. Petit used iron tubes 0.43 inch in diameter and 3 to 23 feet long, set in holes 2.6 inches in diameter. In order to have a uniform exposure of 3.164 square feet of coal surface in all tests, he placed each tube 6 inches from the bottom of the hole in coal and made the space free from tamping 3 feet $3\frac{1}{2}$ inches long. The number of holes tested was 135 and pressures were taken for every meter (3.28 feet).

RESULTS OF TESTS.

In general it was found that the pressure and gas volume increased with depth of holes and with time, but at irregular rates. In the shallow holes up to 10 feet deep the pressures were as low as one-half pound to the square inch, but in some the pressure was as high as $5\frac{1}{2}$ pounds. In holes 23 feet deep the pressure varied from 3 to 18 pounds, but in some of them it was as high as 44 pounds. The time required to give maximum pressures varied. For holes of the same depth the pressure was much less in stalls than in headings.

To test the drainage possibilities of a hole, a boring 15 feet deep with an initial gas pressure of 44 pounds to the square inch was left open for six weeks. The pressure was reduced only to 31.3 pounds, indicating that a hole is not very efficacious in drainage of gas. In one series of tests the holes were bored in the face of a working heading as it progressed into the solid. In one hole of the series a pressure of 17 pounds was observed, although the pressures at previous stages

^a Petit, M. P., Conférence sur la pression du grisou renfermé dans la 13^e couche du Puits du Treuil No. 2, de la Société des Houillères de Saint Étienne: Bull. Soc. de l'Indus. min., ser. 3, vol. 8, 1894, pp. 737-770. Abstracted in Col. Guard., vol. 70, Oct. 15, 1895, pp. 733-734.

of progress had been less than 1 pound. However, adjoining holes in the same heading gave low pressures, 3 pounds or less, which indicated that the high pressure was due to a crevice leading back into the more solid coal. This interesting experiment was repeated with similar results.

CONCLUSIONS.

Petit's conclusions were as follows:

1. High pressure is manifested where coal is exceptionally compact.
2. As a rule pressure in coal for the first 3 feet from the face is slight as the gas is draining off rapidly by cracks caused by blasting and the heaving of the strata.
3. Pressure and volume increase with depth, but irregularly, the rate depending on the compactness of the coal.
4. Districts of high pressure are very irregular in extent because the permeability of coal is variable.
5. The time for equal pressures varies, the main factor being permeability until the final equilibrium is established.
6. Holes bored ahead of workings will not greatly decrease pressure nor drain off much gas.

GAS PRESSURE IN COAL IN AUSTRIAN MINES.

The Austrian commission^a made a number of tests of gas pressure in coal and found in general a rapid increase as the test holes were made deeper and deeper. However, in a 24½-foot hole at Rossitz the pressure was 82 pounds to the square inch, and at Karwin Ostrau a 21½-foot hole gave a pressure of 142½ pounds.

Behrens^b cites a 13-foot bore hole in a newly opened section of the Hibernia mine in Westphalia in which the gas showed a pressure of 218 pounds to the square inch.

VOLUME OF GAS FROM HOLES IN SOLID COAL.

TESTS AT ST. ÉTIENNE.

Petit^c made some tests to determine the volumes of gas given off by holes bored into solid coal in connection with his pressure tests at St. Étienne, France. He bored many holes 3 to 23 feet deep and fitted them with iron pipes securely tamped in such position as to leave 3.164 square feet of coal exposed at the bottom of the hole. In general the pressure and volume of gas increased with depth. Although the larger volumes of gas came with the higher pressures, the relation of pressure to volume was by no means uniform. In two holes of the same depth the gas had about 9 pounds pressure in each,

^a Report of Austrian Fire Damp Commission: Abstracted by *Annales des mines*, ser. 9, vol. 1, 1892, pp. 243-244; *Trans. Fed. Inst. Min. Eng.*, vol. 3, 1891-1892, p. 533.

^b Behrens, —, *Beiträge zur Schlagwetterfrage*: *Oest. Zeitschr. f. Berg- Hüttenwesen*, vol. 45, 1897, pp. 43-46, 63-67, 74-79.

^c Petit, M. P., *Conférence sur la pression du grisou renfermé dans la 13^e couche du Puits de Treuil No. 2 de la Société des Houillères de St. Étienne*: *Bull. Soc. l'indus. min.*, ser. 3, vol. 8, 1894, pp. 737-770; abstracted in *Col. Guard.*, vol. 70, Oct 18, 1895, pp. 733-734.

but one gave off twice as much gas as the other. In 23-foot holes in stalls, with pressure averaging not quite 1 pound to the square inch, the volume of gas was nearly 0.09 cubic foot per minute; with mean pressures of 3 pounds, it was nearly 0.1 foot; and with pressures of 8 pounds, 0.23 cubic foot. In 23-foot holes in headings with a pressure of 7 pounds to the square inch the volume of gas was 0.06 cubic foot; with a pressure of 9.3 pounds, 0.1 cubic foot; and with high pressures, such as 20 pounds, 0.28 cubic foot per minute.

TESTS AT LIÉVIN.

In Simon's^a determination of pressures in solid coal at Liévin, France, measurements were made of the volume of gas given off in holes 30 to 39 feet deep. One hole containing about 13 feet of tamping and an exposed coal area of about 16 square feet, gave off 7½ cubic feet of gas in the first 20 seconds. The volume decreased to about one-third of a cubic foot an hour. Three days later, when the pressure was 42 pounds, gas was given off at a rate of 4½ cubic feet in the first 20 seconds after the cock had been opened, and then the average volume decreased to about one-fifth of a cubic foot an hour, a moderate amount. The volume of gas from a given pressure was in proportion to the vacant space at the bottom of the hole. In tests in another bed of coal (Alfred) one hole with only 1½ square feet of coal exposed at the bottom gave off 0.35 cubic foot of gas a minute, a relative volume fifty times greater than given by the hole in the previous test, the area of coal exposed being considered.

TESTS BY WOOD IN ENGLISH MINES.

Wood,^b in conducting his determinations of gas pressures in the solid coal, measured the volumes of gas given off by some of the holes. Each pipe was set so as to leave a tubular chamber at the end of the hole. Unfortunately the chambers varied in size from 1½ to 3 inches in diameter and from 2 to 6 feet in length, so that the coal surface exposed varied from 95 to 570 square inches. The gas emanations varied from 0.6 to 15.72 cubic feet an hour, or from 0.3 to nearly 6 cubic feet to each square foot of coal exposed. The results were as follows:

^a Simon, A., Notes sur quelques expériences faites au siège No. 1 de mines de Liévin: *Annales des mines*, ser. 9, vol. 8, 1895, pp. 219-231. Abstracted in *Col. Guard.*, Dec. 20, 1895, p. 1164.

^b Wood, L., Experiments showing the pressure of gas in solid coal: *Trans. North of England Inst. Min. Eng.*, vol. 30, 1880-81, pp. 163-256.

Quantities of gas from test holes in solid coal.

Colliery.	No. of hole.	Depth of hole.	Area of coal surface exposed.	Maximum volume of gas per hour.	Maximum pressure.	Colliery.	No. of hole.	Depth of hole.	Area of coal surface exposed.	Maximum volume of gas per hour.	Maximum pressure.
		<i>Feet.</i>	<i>Sq. ft.</i>	<i>Cu. ft.</i>	<i>Cu. ft. per sq. ft.</i>			<i>Feet.</i>	<i>Sq. ft.</i>	<i>Cu. ft.</i>	<i>Cu. ft. per sq. ft.</i>
Eppleton.	1	24½	2.65	15.7	6.0	Boldon...	4	23½	2.60	2.6	1.0
Do....	2	37	3.96	4.3	1.1	Do....	5	28	2.65	10.3	3.9
Do....	3	47	3.96	6.1	1.5	Harton...	1	16	2.40	1.1	.4
Boldon...	1	19	4.24	1.2	.3	Do....	2	27½	2.80	2.3	.8
Do....	2	7½	1.23	.6	.5	Do....	3	37½	3.52	5.0	1.4
Do....	3	32	2.80	.7	.3						
					<i>Lbs.</i>						<i>Lbs.</i>
					204						381
					223						179
					235						197
					425						231
					268						295
					461						

In most cases the minimum amount of gas given off, so far as observed, was about half as much as the maximum. In the Boldon holes the maximum volume was not attained until the measurements had continued for some time, whereas in the others it was quickly attained. The maximum pressure was developed prior to the time of maximum gas emanation. Barometric observations made throughout showed no relation of barometric pressure either to volume or pressure of gas. In No. 3 hole at Eppleton volume observations were made for three weeks, the volume determined varying from 11.40 cubic feet per hour at the outset to 6.86 cubic feet at the end of the test.

EFFECT OF ROCK AND WATER PRESSURE ON GAS PRESSURE IN COAL.

The gas in coal bears the pressure of the overlying strata, so far as that pressure tends to compress the pores holding the gas. Another important factor that has been mentioned is the pressure of water in the overlying strata. It is a well-established fact that all rocks of the upper few thousand feet of the earth's crust contain more or less water in their pores and this ordinarily constitutes a water column extending up to the ground-water level of the district. There are no "impermeable" rocks, although the texture of glassy lavas and of some compact crystalline rocks is so fine as greatly to impede water movement except where the rocks are fissured. Clays and shales permit little water movement, but underground their pores are usually saturated with water. So-called "bone-dry" bore holes and mine workings yield materials that contain considerable interstitial moisture when tested in the laboratory.

The coal measures consist largely of alternations of sandstone and clays or shales containing considerable water that is much in evidence in some mines, and coal itself contains from 1 to 10 per cent of water (air-drying loss). This water, when it fills the pores

and crevices of the coal and extends to the surface, constitutes a column which in a mine 1,000 feet deep has a pressure of 435 pounds to the square inch. This pressure is not manifested directly in a mine, for it is largely sustained by friction, capillarity, or slowness of delivery from the less permeable beds. Although the water may not be sufficiently mobile to penetrate into all the pores in coal containing gas, its pressure must bear on many of them. In a mine, part of the water pressure is relieved by the flow of water from various outlets, but the pressure is often manifested to the amount of many pounds to the square inch when a large water-bearing fissure is encountered. Draining the strata by pumping and mining diminishes the water pressure, but the practical effect of such relief as a factor in gas emanation has not been ascertained.

Rock pressure in mines increases directly with the depth, and averages about 1 pound to the square inch for each foot of depth. Therefore coal at a depth of 1,000 feet is under pressure of a half ton to the square inch. The bearing of this factor on gas in the pores of the coal has not been fully considered, but when the coal is removed and released from such a great compressive strain it expands materially and there is readjustment of stress from the interior to the surface of each lump of coal, as well as of the face, of the pillars, and of the roof and floor.

PERMEABILITY OF ROCKS AND COAL TO GAS.

The gas in the coal, especially where the strata are uplifted and flexed, is only a remnant of the original amount, for leakage has been in progress for thousands of centuries. That gas will penetrate rock for long distances has been well established, and although sheets of fine-grained materials must greatly impede its circulation, yet time is by far the most important factor. Many cases are known of gas wells securely plugged from which the gas has flowed laterally into other wells or into mines. The rate at which gas escapes to the surface in any coal field is as a rule very slow and the escape is so general that it is not apparent, except in places along faults or joint plains where at some localities it manifests itself most vigorously.

ESCAPE OF GAS FROM STRATA NEAR WILKES-BARRE, PA.

Not long ago a large amount of coal gas was bubbling up through the water of the Susquehanna River and the sloughs opposite Wilkes-Barre, Pa., and it was frequently ignited. At one place a pipe forced some distance into the sand where bubbling occurred supplied gas enough to light and heat a house near Market Street bridge.

Several years ago a vigorous "blower" was uncovered in the deep cut of the Nanticoke Branch of the Central Railroad of New Jersey

a mile east of Nanticoke, Pa., which as yet shows little diminution of force. There are in various parts of the field many small gas outlets that illustrate how general is the escape of gas.

The proportion of gas that has escaped from coal beds can not be determined, but it varies greatly, and in places the gas is practically all gone. This is the case along the outcrop zones, and in shallow basins where the cover is thin or consists of coarse-grained rocks. As shown in previous pages probably all coal contains some methane, but the word "gaseous" is only applied when there is a sufficient emanation to show in a safety lamp.

ESCAPE OF AIR FROM SEALED CHAMBER IN MARYLAND MINE.

An interesting observation in this connection was made some years ago by Randolph^a in a mine in the Georges Creek district, Md., working the Pittsburgh bed, which is there soft and friable and intersected by numerous slips. During the installation of a new pumping system the water was allowed to flood the lower workings and as some of these chambers rose from the main gangway and had no outlets, a large body of air was entrapped in them under a water head of about 40 feet part of the time. At the end of 18 months, when the water was pumped out, it was found that the chambers had been filled with water to the roof and consequently all the air had been forced out through the nearest outlet which was 200 feet away through the coal bed. When the mine was unwatered below the level of the chambers some of the air worked back again and finally presented a partial vacuum equal to that produced by a water column 7 feet high.

Randolph also cites an instance of the impenetrability of the "slate" above the coal in this mine. A heading passing over a dome was partly filled with water that imprisoned considerable air. When the mine was pumped out it was found that the water level had never reached the roof, although at one time the air had supported a head of upward of 80 feet of water. In order to escape the air had to pass through the "slate" at right angles to the stratification, but could not do so to any perceptible degree.

The slowness of movement of gas through the coal has been demonstrated by some of the pressure tests described on page 60, which were so conducted as to throw light on this condition. However, the demonstration applied mostly to solid coal and the time covered by the experiments was short.

Simon found evidence of the passage of gas into the overlying strata at Liévin, France.

^a Randolph, B. S., Notes on unwatering a flooded mine and on permeability of natural strata to air: Trans. Am. Inst. Min. Eng., vol. 24, 1894, pp. 21-25.

GAS IN STRATA ADJOINING THE COAL.

Shales and sandstones above or below the coal bed or interlaminated with it contain more or less gas which is liberated as mining progresses.

Morin,^a who investigated the gas conditions at Liévin, believes that such beds are an important source of gas in mines and the results of some of his tests appear to substantiate the idea. It is found that generally the amount of gas in a mine increases as the mining area is extended, even if the area of working faces does not increase. This increase is due in large part to increased area of coal exposed in rib, pillar, and gob, but Morin believes that these soon lose their gas and the increment comes from the floor and roof. As coal is removed and for a while afterwards the overlying and underlying rocks move more or less and thus many fissures are formed from which gas escapes either from the rock itself or from adjoining coal or coaly beds. Naturally this emission of gas is more rapid near the working faces when there is more gas and fissuring is most active. Bore holes drilled into the floor and roof of the coal at Liévin showed little gas at a distance back from the face, although there might be considerable of it above and below the solid coal.

Simon^b tamped 20-foot tubes in the roof of the Frederic coal bed at the Liévin mine and found a pressure of only 1.4 pounds per square inch, though the pressure in the coal below was from 62 to 80 pounds. In the newly opened Alfred bed the returns at working faces contained 0.4 per cent of methane, whereas at the extremity of the same return the amount was nearly 0.9 per cent and under similar conditions 0.5 to 0.9 per cent has been found in the Arago bed.

A further example is given in an airway passing above the Du Souich bed. It showed no gas even when the ventilation was not effective, but when two working chambers in the coal below were extended under this airway gas became perceptible which undoubtedly worked up through fractures in the strata, due to the removal of coal.

Morin also made tests of the amount and the pressure of gas in rocks. One test was in an overturned part of the Alfred bed, where a 5-foot hole in the wall gave a pressure equal to 0.4 inch of water (about one-seventieth of a pound to the square inch). A second hole in a gangway 2 years old yielded gas containing 6.3 to 5.7 per cent of methane, the volume being one-eighth to one-fifth of a cubic inch a minute. Similar tests in the Du Souich workings at different places

^a Morin, Léon, De l'influence des variations de la pression atmosphérique sur les dégagements de grisou: *Annales des mines*, ser. 10, vol. 16, 1909, pp. 385-437. Abstracted in *Eng. and Min. Jour.*, vol. 90, Sept. 17, 1910, pp. 565-568.

^b Simon, A. Notes sur quelques expériences faites au siège No. 1 de mines de Liévin: *Annales des mines*, ser. 9, vol. 8, 1895, pp. 226-227.

gave varying results. One hole 5 feet deep in the wall of some workings 3 years old gave off air and gas containing 2.2 per cent of methane, but no pressure was evident; another hole in the roof at another point in workings a year old gave off gas containing only 0.39 per cent methane, but doubtless this hole had crossed a fissure; and a third hole in the roof gave off a small flow of gas containing 5 and 4.6 per cent of methane. These tests all prove that methane lingers in adjoining strata after the coal has been removed. Some tests were made in undeveloped sections, one being 65 feet above the Arago bed. A hole 6 feet deep yielded samples of gas containing only 0.39 and 0.065 per cent of methane. Another hole 5 feet deep in an unworked region yielded air containing 0.19 per cent methane, whereas a third hole 4 feet deep, 13 feet above the Beaumont bed, yielded gas with a methane content of 3 per cent. These results show that at a distance from the coal the amount of methane in the strata is very small, but near the coal the amount is considerable.

In one shaft in Prussia gas began with clay "slate" at 13 feet below ground and extended to the top of the Wealden sandstone 551 feet below. The gas outflow was very strong at 515 feet in dark bituminous clay "slate." In sinking the shaft (No. 3) for the Woodward mine just north of Wilkes-Barre, Pa., gas was found in the sandstones and other rocks in large quantity and gave trouble all the way down. At one stage of progress the outflow was estimated at 1,000 cubic feet a minute. Possibly the gas was rising from the underlying coal beds, but some of the rocks were shales, relatively impervious.

A boring in the Schaumburg district was in shales; at a depth of 695 feet, or 280 feet above the coal bed, gas was encountered which threw a 2-foot column of water for one and one-half hours and was in vigorous action at times for two and one-half months.

GAS IN A SANDSTONE STRATUM IN PENNSYLVANIA.

A rock tunnel in the "Harry E" anthracite mine near Wilkes-Barre, Pa., penetrated a sandstone that gave off considerable methane. A sample of the fresh rock was sealed and sent to R. T. Chamberlin for a determination of its contained gases. The sample was broken into fragments the size of small marbles, which were then placed in a vacuum bottle and the air exhausted. Of course, considerable gas was lost in this process. At the end of 21 days the gas that had accumulated in the vacuum bottle during this time was pumped out and analyzed. The volume of gas at 32° F. and 30 inches pressure, amounted to only 0.013 per cent of the volume of the rock used. The composition of the gas was as follows:

Composition of gas from sandstone layer.

	[Analyst, R. T. Chamberlin.]	Per cent.
Methane.....		4.2
Carbon dioxide.....		1.9
Air.....		70
Nitrogen (excess over air).....		23.9
		100

It is probable that the methane in this rock came from the coal below and was stored in the pores of the sandstone. Doubtless most of it passed out of the rock rapidly, as the excavation progressed.

RELATION OF VOLUME OF GAS TO UNDERGROUND TEMPERATURE.

Very few tests have been made to determine the relation of gas emanation to underground temperature. At all places temperature increases with depth, but the rate of increase is variable. Higher temperatures naturally cause gas to escape from the coal more rapidly, but whether it is sufficient to be of any practical moment has not been determined.

COAL OXIDATION AS A SOURCE OF GAS.

A possible factor in the mine-gas problem is the abstraction of oxygen from the air by coal. This process appears to progress in all kinds of coal, but at very different rates, some coals showing slight disposition to absorb oxygen, whereas others absorb a large amount.

INCREASE IN TEMPERATURE FROM OXIDATION.

The result of oxidation is the production of carbon dioxide, accompanied by an increase of temperature which accelerates the emission of the methane held in the coal. Haldane and Meachem^a investigated this matter and found that the resulting increase of temperature was notable. At the Hamstead colliery in South Staffordshire at the bottom of the shaft, which is 1,880 feet deep, the average air temperature was about 11° higher than at the surface and rose 6° F. for every 3,000 feet along the main intake, rising to 80° or 85° F. in the workings. The temperature of the return air diminished from the face to the upcast, but at a slower rate than the increase in the intake air. In one split the temperature of the intake air was 60° F., of the rock in the mine due to depth, 68° F.; and of the upcast air, 77° F. The ventilating air lost in oxygen 3.13 times more than it gained in carbon dioxide in this mine, but in other mines the ratio was only about 1.6.

^a Haldane, John, and Meachem, F. G., Relation of underground temperature and spontaneous fires in the coal to oxidation: Trans. Inst. Min. Eng.; vol. 16, 1899, pp. 457-462.

Haldane and Meachem concluded that the increase of temperature was mostly due to oxidation of coal, and that in general "the temperature increment of the mine air rose with the diminution of oxygen." They also suggest that carbon dioxide in various kinds of mines and pits may be largely due to oxidation. A self-recording maximum thermometer placed in a 10-foot hole in the gangway rib soon recorded 66° F., but four years later the temperature had risen to 90° F. This and other observations show the gradual penetration of oxidation and the resulting rise of temperature in the coal exposed.

ABSORPTION OF OXYGEN BY COAL SAMPLES.

Porter and Ovitz^a found that oxygen is absorbed rapidly and for a long period by fresh coal and but little carbon dioxide results. An excellent illustration was a test with 22 pounds of Connellsville coal. In one day after mining it absorbed nearly 60 cubic inches, or half of the oxygen in 610 cubic inches of air, and gave off little more than one-tenth as much carbon dioxide as would have been formed if all the oxygen absorbed had combined with carbon and been evolved as carbon dioxide. They suggest that the reaction is probably a direct combination of the oxygen with certain fixed components of the coal, as it has been shown by Boudouard^b and others that certain compounds in coal are unsaturated with oxygen and its absorption produces humic acid or related substances.

The rapidity of absorption was shown by tests with lignite from Sheridan, Wyo., and bituminous coal from Benton, Ill., which in less than 15 days had exhausted oxygen from a volume of air equal to their own volume, but neither coal gave off more carbon dioxide than one-tenth of the volume of oxygen absorbed. That this oxygen is probably taken into chemical combination rather than mechanical absorption is shown by a test of Benton coal. A sample of this coal, that had been supplied with oxygen for five months, absorbed nearly seven times as much oxygen as it originally contained. The sample was heated to 212° F. for 15 minutes. The gases evolved contained no excess of oxygen over that in air.

EFFECTS OF VARIATION OF ATMOSPHERIC PRESSURE ON GAS EMANATION.

It is a very old and deeply seated idea that there is a close connection between the weather and the amount of fire damp in mines, and even before the barometer was invented coal miners regarded increased gas emission as an indication of the close approach of stormy

^a Porter, H. C., and Ovitz, F. K., *The escape of gas from coal*: Technical Paper 2, Bureau of Mines, 1911, pp. 10-12.

^b Boudouard, O., *Études sur les charbons*: Bull. Soc. chim. de France, ser. 4, vol. 5, 1909, pp. 365-381.

weather. The idea has been prevalent also that explosions are most frequent at times of low atmospheric pressure or storms. Sir Frederick Abel has shown this opinion to be largely erroneous. He gave a list of explosions during the years 1875 to 1885, involving the loss of 2,229 lives, and by reference to weather records showed that only 17.4 per cent of the mortality was at a time when the barometric pressure was below the average and that half of the explosions occurred when the pressure was increasing. If the frequent small accidents due to various causes be excluded, three out of four of the explosions were at times of high pressure or anticyclonic conditions. We now know, furthermore, that many of these explosions were caused by dust ignited by some accidental explosion and may have had no relation to increase of gas in the mine.

It is generally conceded that diminution of atmospheric pressure affects the liberation of gas from coal, especially from old workings and crevices where the gas has accumulated. This relation, the effects of which have been repeatedly noticed by miners, has been fully substantiated by elaborate scientific investigations. In places where most of the escaping gas has been under notable pressure in the coal the volume emitted can not be materially affected by a change of atmospheric pressure. Various observers have shown that as gas approaches a face of coal its pressure gradually diminishes and in the outer few inches becomes very low. A "fall of barometer" of 1 inch indicates a diminution of atmospheric pressure of nearly one-half pound per square inch. Such a change taking place in a few hours must decidedly affect the equilibrium of the atmosphere in the mine. The chief result is an expansion of the air, a decrease of one-half pound in pressure causing an expansion in volume of about one-thirtieth, or $3\frac{1}{4}$ per cent. As the expansion is gradual, the results can have no direct effect on the strong air currents of the main ventilation, but the still air in the old workings and in crevices would be affected by it and some gas held in the coal at the pressure of the atmosphere before the change would be given off when the pressure was relieved. An air current traveling 500 feet a minute, a not unusual rate in some gaseous mines, has a pressure of only about 1 ounce to the square inch, equivalent to about 0.14 inch of barometric change.

The volume of gas that may be forced out of old workings with changes of atmospheric pressure may be more clearly understood if considered on a quantitative basis. Assume that an area of old workings 100 acres in extent and 10 feet high is filled with still air carrying 2 per cent methane and having outlet to an air current of 20,000 cubic feet a minute. If there were 50 per cent of open space the volume of methane in these workings would be 440,000 cubic feet. As a barometric fall of 1 inch causes an expansion of about 3 per cent in a given volume of air, a decrease of atmospheric pressure

of 0.1 inch would expand this still air 0.3 per cent and drive 1,320 cubic feet of methane into the airway. If one hour were required for the barometric change and reestablishment of atmospheric equilibrium, the rate of discharge would be about 22 feet a minute, which would add to the gas in the airway only 0.11 per cent of methane. However, the change might disturb the general equilibrium and start the flow of the entire body of still air in the old workings into the return. In the 20 hours or more that would be required to drain the 100 acres at the rate of 20,000 cubic feet a minute, the methane increment in the air current would be 1.8 per cent, which would of course be a most decided change, whatever the normal methane content of the return air.

In many mines there are vast areas of old workings in part not well ventilated, and the air in these contains more or less methane. The Prussian Fire Damp Commission ^a found that in the Ath-Gouley mine (Wurm coal field) the spaces amounted to 9,110,000 cubic feet, and 7,410,000 in part of the Gemeinschaft mine. In one English mine it is estimated that the open space in old workings amounts to 36,000,000 cubic feet.

In the following paragraphs are given the results of some of the many investigations of the relation between atmospheric changes and variations in methane in mines.

INVESTIGATIONS IN BRITISH MINES.

Galloway's ^b observations in Scotland did much to establish the belief that there was an important relation between barometric change and the volume of gas given off by mines, and although his investigation was not extensive it led to the passage of the Mines Regulation Act in 1872, which required barometric observations at all collieries. Experiments in German mines by Schöndorff ^c and Nasse ^d in 1875 to 1877 showed that the methane content in the returns was greatly increased in periods of low pressure, especially in those mines where there were extensive areas of old workings with poor ventilation.

^a Report of Prussian Fire Damp Commission (translation): Trans. Fed. Inst. Min. Eng., vol. 4, 1898, p. 643.

^b Scott, R. H., and Galloway, William, On the connection between explosions in collieries and weather: Proc. Roy. Soc., vol. 20, 1872, pp. 292-305.

^c Schöndorff, A., Untersuchung der ausziehenden Wetterströme in den Steinkohlen-Bergwerken des Saarbrückens: Zeitschr. Berg.-Hütt.-Salinenwesen, 1876, p. 114.

^d Nasse, V. H., Beobachtungen über die Beziehungen des Auftretens schlagender Wetter in Steinkohlengruben zu den Veränderungen des Luftdruckes: Zeitschr. Berg.-Hütt.-Salinenwesen, vol. 25, 1877, pp. 267-280.

The inquiry into the ventilation of coal mines in England in 1908 by Cadman and Whalley ^a of the Royal Commission of Mines resulted in finding that in many mines a diminution of atmospheric pressure was reputed to have caused more mine gas to appear in the workings, but no precise determinations were made. Redmayne,^b chief inspector of mines of Great Britain, states that sudden diminution of atmospheric pressure causes large amounts of gas to come out of old workings, and also increases the discharge of blowers. Wabner ^c discusses the influence of atmospheric pressure and gives various data showing its importance.

INVESTIGATIONS IN PRUSSIAN MINES.

TESTS BY HILT NEAR AIX-LA-CHAPELLE AND KARWIN.

The Prussian commission investigated the effect of atmospheric influence on methane emanation at the Ath-Gouley and Gemeinschaft mines near Aix-la-Chapelle and at the Gabriele mine near Karwin, in Austrian Silesia. At all three places a close relation was found, so that the results were confirmatory.

The investigations at the Ath-Gouley and Gemeinschaft mines were conducted in September and October, 1885, by Hilt.^d Both mines work the Grosslangenberg bed, which is very gaseous. The Ath-Gouley mine contains large areas of old workings, but the Gemeinschaft mine was in fresh dip workings which gave off much gas. In each mine a split of the return air was sampled every morning at 5 o'clock, and the volumes of methane and carbon dioxide were determined. One week the samples were taken four times a day, and for another week samples were taken in two different parts of the Gemeinschaft mine where the workings were particularly fresh. Air from workings in another bed, "Meister," was included in this last series. These latter samples were taken to determine whether the variations in mining from day to day and during parts of the day would account for the changes of methane content in the return, but the relation was not established. Some of the results of the tests in September, 1885, are given in figure 10.

^a Cadman, John, and Walley, E. R., Report of an inquiry into ventilation of coal mines and methods of examining for fire damp, 1909.

^b Redmayne, R. A. S., Ventilation of mines, London, 1911.

^c Wabner, Robert, Ventilation in mines, 1903.

^d Hilt, C., Bericht über Versuche betreffend den Einfluss des wechselnden Luftdruckes auf die Entwicklung des Grubengases. Im Auftrage der Preussischen Schlagwetter Commission. Zeitschr. Berg.-Hütt.-Salinenwesen, vol. 24, 1886, pp. 72-90. Translated and reviewed by M. Simon, Bull. Soc. l'Indus. min., ser. 3, vol. 1, 1887, pp. 595-625; by M. de Vaux, Rev. univ. des mines, vol. 19, 1887, pp. 395-419; and by L. L. Belinfaut, Trans. Fed. Inst. Min. Eng., vol. 4, 1893, pp. 645-646.

Hilt thought that the experiments proved absolutely that variations in atmospheric pressure affected the emission of gas in both mines. The barometric variations were small at the time of the tests, but their effect was plainly discernible in corresponding variations in methane content, especially in the returns of the Ath-Gouley mine. A rise in barometer was followed by diminution of the volume of methane and carbon dioxide, and a fall in barometer by an increase in the volume of methane, and always within 24 hours. The more rapid the rise or fall the greater was the variation in the percentage of gas. Moreover, the outflow of methane showed a tendency toward equilibrium, for when, by reason of a considerable fall in barometric pressure, the flow had risen much above the mean, it appeared to have

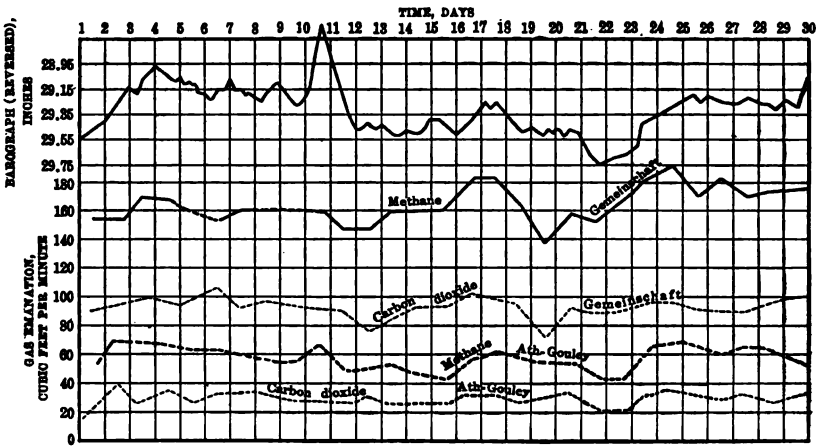


FIGURE 10.—Curves showing variations in volume of methane and carbon dioxide in relation to variations in atmospheric pressure, Gemeinshaft and Ath-Gouley mines, Prussia, September, 1885. After Hilt.

a tendency to decrease. The absolute pressure was found to be of less importance than the relative changes, especially when the changes were rapid. In tests made at the Ath-Gouley pits, October 11 to 17, in a region of old workings having open spaces aggregating 9,110,000 cubic feet, comparatively slight variation of atmospheric pressure caused variations of 70 per cent in the mean volume of gas in the returns, and these variations appeared to be independent of the amount of coal being mined. The opinion was stated that at this mine a great decrease of atmospheric pressure might double the volume of methane. In the long-wall and dip workings of the Gemeinshaft mine, where most of the gas was from solid coal, the effect of changes of atmospheric pressure was much less marked and it was somewhat modified by the variations in the mining operations. One

notable irregularity in this mine was a great increase of methane on October 21 and 22, with only a slight fall of barometer, and as there were no old workings this increased flow must have come from the face. Doubtless some crevices were opened or a body of coal entered which contained an increased volume of gas.

REVIEW OF HILT'S WORK BY MALLARD AND LE CHATELIER.

Mallard and Le Chatelier ^a reviewed Hilt's report in great detail, and, although they agreed that some relation was proven, they claimed that it was in returns receiving air from extensive areas of poorly ventilated old workings. They thought, however, that the tests in the new workings appeared not to show any relation, for there was more discordance than accordance in the relations in the fresher workings at Gemeinschaft and not very close relation in the workings in the Meister bed, notably in one of the most rapid falls of barometer. They point out that this is to be expected in fresh workings where the gas is given off under pressure. Mayer,^b who made extensive researches on gas evolution in mines in Polnisch-Ostrau, concurs in the idea of increased emanation of methane caused by diminished atmospheric pressure, but believes that old workings are principally affected and in no case is the increase likely to be dangerous in a well-ventilated mine.

INVESTIGATIONS IN AUSTRIAN MINES.

TESTS BY KOEHLER AT KARWIN, AACHEN, AND HIBERNIA.

A series of tests extending over a period of six months was made in 1885 by Koehler ^c in the Gabrielle mine at Karwin, in Westphalia. The mine is a very gaseous one, with an average of about 2½ per cent of methane in the main return, equal to about 800 cubic feet a minute. Five-gallon samples of this air were taken daily and also samples of the air in a return from districts entirely in fresh workings in the Karl bed. By plotting percentages of methane under a curve of atmospheric pressures based on readings at the top and the foot of the shaft, he obtained the results given in figure 11. He found that the proportion of methane increased with barometric fall and that the more abrupt the fall the greater was the rate of increase.

^a Mallard and Le Chatelier, Sur les travaux de la Commission Prussienne du grisou: *Annales des mines*, ser. 3, vol. 9, 1886, pp. 638-664.

^b Mayer, Joh., Über den Einfluss der Luftdruckschwankungen auf die Entwicklung von Schlagwettern bei besonderer Betrachtung der auf Gabrielen-Zeche in Karwin ausgeführten Versuche; *Oest. Zeitschr. Berg- Hüttenwesen*, vol. 34, 1886, pp. 35-38, 53-61, 69-72.

^c Koehler, G., De l'influence des variations de la pression atmosphérique sur les dégagements de grisou. Rapport sur les expériences faites dans les mines de Karwin (Silésie autrichienne). Translated by Grey, M. R., *Bull. Soc. l'indus. min.*, ser. 3, vol. 1, 1887, pp. 595-625; and *Rev. univ. des mines*, vol. 19, 1886; reviewed by Shorn, G., *Ann. trav. pub. Belgique*, vol. 4, 1886, pp. 453-495.

He confirmed Hilt's observation that the amount of methane does not depend upon the absolute pressure but on the rapidity of change. If after a rapid diminution of pressure, there was little or no further change, there was only a gradual increase of methane, whereas if after a rapid increase of pressure there was little or no barometric change, there was a slight decrease of methane content. Therefore the time of maximum gas content did not precisely coincide with that of minimum pressure. The curves of barometric pressure and volume of methane showed 105 concordances and 85 discordances.

Koehler made an important test to determine the effect of artificial rarefaction of the air on methane emanation. The intake was sealed while the upcast fan continued to run so as to diminish the air pressure in the mine, which amounted to 2.5 millimeters, or about one-tenth inch. The results were as shown in the table on the following page.

As this mine contained large areas of old workings the great increase of methane in the air from the main upcast may have been in large part derived from them. The workings in the Karl bed, however, were all fresh, and the increase of methane from them was probably due to increased emanation from the solid coal when the atmospheric pressure was diminished. There are some possibilities of error in these diminished-pressure observations because air volumes are not very accurately measureable with anemometers running slowly.

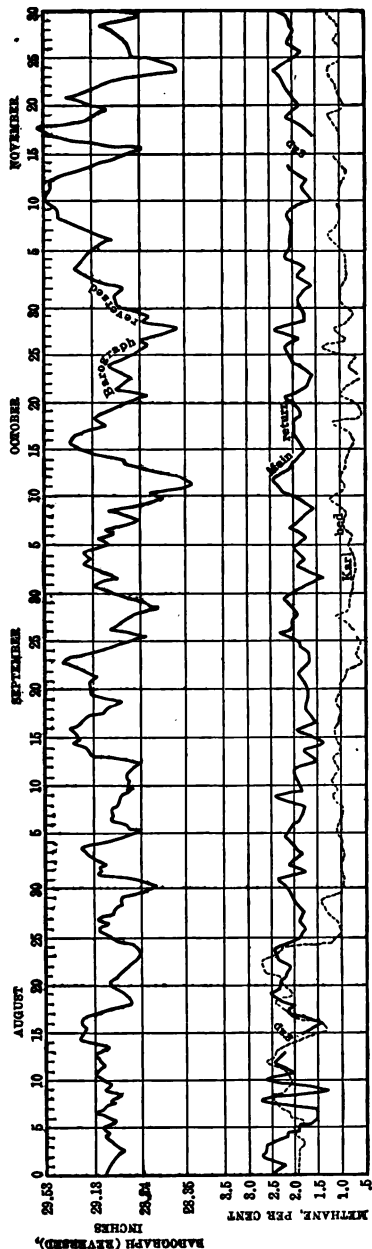


FIGURE 11.—Curves showing variations in percentage of methane and barometric pressure in the Gabriele mine, Karwin, Austria. Full line represents samples collected in main return; dotted line represents samples of return air from workings in Karl bed.

Variation in volume of methane in return airways of Karwin mine under diminished air pressure.

Number of test.	Fall of mercury in barometric pressure.	Point where sample was taken.	Volume of methane at beginning of test.	Volume of methane at conclusion of test.	Increase in methane content.
	<i>Inches.</i>		<i>Cubic feet.</i>	<i>Cubic feet.</i>	<i>Per cent.</i>
First.....	0.10	Upcast.....	710	1,301	80
Do.....	.10	Karl bed.....	146	206	41
Second.....	.08	Upcast.....	725	1,079	50
Do.....	.08	Karl bed.....	190	228	20

In 1888 Koehler^a made another series of similar observations in the Kohlscheid mines near Aachen.

The barometer pressures were taken at noon and midnight and the amount of gas was determined morning and night. A gasometer installed behind the fan of the ventilating shaft of the Ath-Gouley mine, and placed in such a manner as exactly to fill itself in 12 hours, furnished average samples of the composition of the air in the mine. For taking the amount of gas a Coquillion fire-damp meter was used, and the analyses furnished by that apparatus were controlled by experiments at the laboratory of the Aachen School of Mines. The comparison of the curves of methane and atmospheric pressure is very instructive. Thus, for January, 1889, during the first fortnight, to the lowest pressure (737 millimeters) corresponded a maximum holding power of 3.9 per cent of methane, which became nil when the pressure returned to 770 millimeters. During the second fortnight the variations of the curves also agreed. In February a fall of pressure showed itself by an increase of methane up to 7 per cent, and ended by disappearing. Then, the pressure rising, the methane decreased. Sometimes considerable barometric falls appeared to be without effect, but, generally speaking, the maximum of pressure coincided with the absence of methane. In summer escapes were much more marked, and the proportion of 8 per cent was even exceeded at the end of July. The phenomena were not simultaneous, one or two days being necessary for the alteration of the subterranean air to attain its maximum, but there were cases observed in which the action was in some way immediate.

CONCLUSIONS BY BEHRENS.

Behrens,^b in a review of the observations made by himself and Koehler at Karwin and Hibernia, Westphalia, states the following conclusions:

(1) Increasing atmospheric pressure retards the escape of fire damp; diminishing pressure accelerates it.

^a Koehler, G., The relations of barometric depression and escapes of fire damp: Eng. and Min. Jour., vol. 5, p. 287, March 7, 1891; abstracted in Col. Guard., vol. 61, 1891, p. 320.

^b Behrens, —, Beiträge zur Schlagwetterfrage: Oest. Zeitschr. Berg-Hüttenwesen, vol. 45, 1897, pp. 43-46, 63-67, 74-79.

(2) The greater the increase in atmospheric pressure per unit of time, so much the more is the liberation of gas diminished; the more the pressure falls per unit of time, the larger is the escape of the gas.

(3) If the barometric pressure, having attained a certain height—the result of which has been to diminish the escape of fire damp—remains at that level for some time or permanently, the percentage of gas gradually increases again, though without attaining the original figure, so long as the increased pressure continues. The diminution in the outflow, induced by a permanent increase in the atmospheric pressure, is less when the gas pressure in the coal is high, and greater when the gas pressure is low. Conversely, when the atmospheric pressure falls—which condition is accompanied by an increase in the escape of gas—and then remains at this point, either for some time or permanently, the percentage of gas gradually decreases again, though it does not recede as low as the original level, so long as the pressure remains constant; the increased outflow of gas, due to the permanent fall in the pressure, is smaller when the gas pressure in the coal is high, and greater when the gas pressure is low.

(4) Should a sudden rise of atmospheric pressure be succeeded by a more gradual increase, a slow acceleration of the gas outflow ensues; on the other hand, when a rapid fall in the barometer is followed by a more gradual one, the retardation of the gas outflow proceeds slowly.

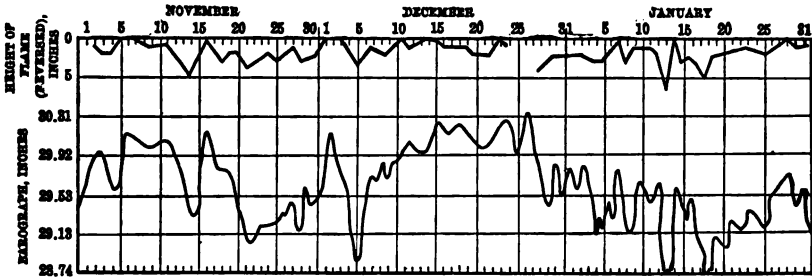


FIGURE 12.—Curves showing relation between length of flame of a blower and variations in atmospheric pressure, November, 1885, to January, 1886, at Hanover mine, Westphalia.

In no case, however, do the maxima or minima of the barometric curves correspond to maximum or minimum rates of outflow of the gas.

TESTS AT HANOVER MINE.

In 1885 Broockman^a made an investigation at Hanover mine No. 2 near Wattenscheid, Westphalia, to determine the effect of variations in atmospheric pressure on the volume of gas from a "blower." The blower was connected to a tube, and the gas burned so a comparison could be made of variations in height of flame with variations in pressure as recorded on a barograph. The results are given in figure 12, which shows a close connection between the two indicated by an increased flow of gas at the time of diminished atmospheric pressure, as in the tests of mine air at Karwin and other places.

Heise and Herbst^b have shown that notable pressures of the gas are obtained only by boring several meters back into the solid, and

^a Broockman, K., Ueber den Einfluss des Luftdruckes auf die ausströmende Gasmenge eines Bläasers, *Zeitschr. Berg-Hutt. Salinenwesen*, vol. 34, 1886, pp. 155-156; *Bull. Soc. l'indus. min.*, ser. 3, vol. 1, 1887, pp. 624-625.

^b *Bergbaukunde*, vol. 1, 1906, pp. 458-461.

that near the face the pressure is only slightly in excess of the air pressure. This refers solely to the pressure of the free gas and not to the gas in the pores of the coal which escapes too slowly to affect an ordinary gage.

INVESTIGATIONS OF THE AUSTRIAN FIRE DAMP COMMISSION.

The Austrian Fire Damp Commission^a made many observations to ascertain the relation between gas emanation and changes of atmospheric pressure. The tests were made simultaneously for four weeks in six districts, some in gaseous mines, others in nongaseous mines, some in mines with old workings, and others in mines without old workings. Samples were taken three times a day and data obtained as to the amount of coal mined, surface of coal exposed, number of men and animals working, and other details. In some of the returns in gaseous mines the samples contained $1\frac{1}{2}$ per cent of methane or 7,469 cubic feet per ton of coal extracted, and in less gaseous mines the methane content in the return air averaged 0.3 per cent, or 519 cubic feet per ton of coal extracted.

In five districts in gaseous mines without old workings no concordance was found between barometric changes and the amount of methane. In mines in a slightly gaseous district, however, the volume of methane varied from $12\frac{1}{2}$ cubic feet a minute when the barometer was low to 11 cubic feet when it was higher than the average. On the other hand, in mines containing extensive old workings the proportion of methane closely followed the changes in atmospheric pressure. The return air in the old workings of the Tiefbau mine was sampled from February 13 to March 5, 1889, with results shown in figure 13.

The effect of barometric changes on gas emanation from very gaseous fresh coal was tested by measuring the volume of methane from bore holes. It was found that the volume produced from these in equal given times showed no relation to the variations of atmospheric pressure. This latter test, however, did not show fairly the conditions at the exposed surface of the coal, where ordinarily most of the gas has been given off and therefore the pressure is far less than in bore holes.

INVESTIGATIONS IN FRENCH MINES.

The French commission created in 1887 to study coal gas^b gave considerable attention to the question of the effect of barometric changes. The commission was doubtful as to there being any material effect or enough to be a factor of danger in properly ventilated mines. Le Chatelier, however, believed that old workings were sus-

^a Atkinson, W. H., The report of the Austrian Fire Damp Commission: Trans. Fed. Inst. Min. Eng., vol. 3, pp. 531, 1891-92, and Chesneau, G., Annales des mines, ser. 9, vol. 1, 1892, pp. 245-248.

^b Goupillière, Haton de la, Rapport présenté au nom de la commission d'étude des moyens propres à prévenir les explosions du grisou: Annales des mines, ser. 7, vol. 18, 1880, pp. 193-415.

ceptible. Chesneau^a made daily tests for 11 months in 1886 at the Herin mine at Anzin at a place where only a small part of the air could come from old workings even with great barometric change. The methane content of this air averaged about 0.8 per cent, or equivalent to an outflow of about 118,570 cubic feet a day. He found 81 days of concordance between gas emission and barometric fall, 46 days of discordance, and 51 days of independence. (See fig. 16, p. 100.) He concluded that for a bed that yields a permanent and relatively regular outflow of gas sudden barometric falls cause a notable increase in the emanation of methane.

TESTS AT LIÉVIN.

In 1907 Morin^b made an extended investigation to ascertain the relation between atmospheric pressure and gas at Liévin, France. A long time previously he had noted such relation, especially in certain galleries not well aired and also along some of the returns.

The tests at Liévin were continued for two months. In shaft 13 the percentage of gas was determined every hour, with the unfor-

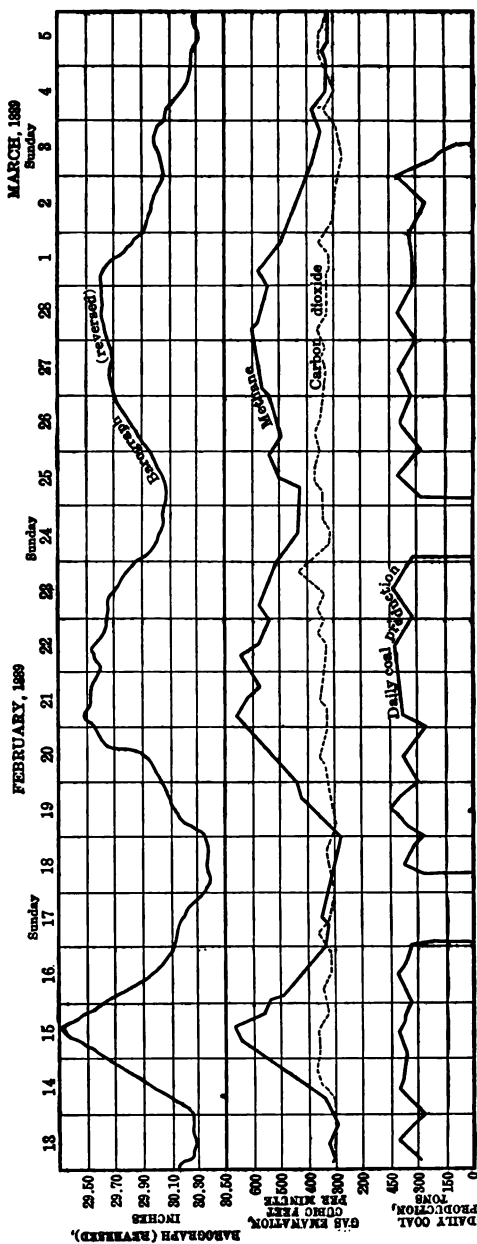


FIGURE 13.—Curves showing variations in volumes of methane and carbon dioxide in return air at Tiefbau shaft, Austria, showing relation of variations in atmospheric pressure and also of amount of coal mined.

^aChesneau, G., *L'influence des mouvements du sol et des variations de la pression atmosphérique sur les dégagements de grisou*: Annales des mines, ser. 8, vol. 13, pp. 359-428.

^bMorin, Léon, *De l'influence des variations de la pression atmosphérique sur les dégagements de grisou*: Annales des mines, vol. 16, ser. 10, 1909, pp. 385-437; abstract in Eng. and Min. Jour., vol. 90, Sept. 17, 1910, p. 565.

tunate exception of Sundays, and in three shafts every eight hours. At shaft 13 the upcast from an area of nearly 200,000 square yards, including extensive worked-out areas, carried 16,800 cubic feet of air a minute with remarkable uniformity. The samples of air were collected by displacement in water-filled bottles. Except for a small area in an underlying bed there were no workings above or below. The principal results are presented in figure 14.

Morin's deductions from these results were as follows:

Even slight variations in atmospheric pressure may induce variations in the escape of methane, and other things being equal, when the atmospheric pressure remains constant the liberation of methane is uniform. When atmospheric pressure increases the escape of gas diminishes, and conversely.

Maximum atmospheric pressure corresponds to minimum percentage of gas and vice versa.

In some tests a decrease of 1 inch in the height of the mercury column increased the proportion of methane 50 per cent.

ATMOSPHERIC CONDITIONS IN OLD WORKINGS.

As the air tested came from a district comprising a large area of old workings, it appeared likely to Morin that the old workings might act as a great reservoir and give a large outflow of gas at times of diminished atmospheric pressure. Even where the old chambers were filled with gob and the surface had settled more or less the amount of free space was large. He found that when certain old workings were pumped out the volume of water when measured showed that the voids amounted to 30 per cent of the original excavation. A careful estimate of voids in workings in the Du Souich bed, workings which covered nearly 200,000 square yards and were nearly 5 feet high, showed a total of 4,028,000 cubic feet. It was found that if differences in the volume of gas coming out of the mine under different pressures were all supposed to be caused by the open workings the volume of pure methane in the voids in the mine, if the voids were assumed to be filled, would range from 107,660 to 10,266,100 cubic feet, the larger volume corresponding to a longer period of time. This result could be explained only by the hypothesis that the gas and air might be in strata of varying composition, those mixtures richest in gas occupying the highest levels. Under such a condition a slight diminution in atmospheric pressure might affect only the mixtures poor in gas occupying the lower levels, whereas further diminution would bring out mixtures containing more gas. However, careful sampling of the air in the old workings did not reveal such a condition, which moreover is not consistent with the laws of diffusion. Therefore Morin concluded that in these experiments the old workings

were not the only cause, nor even the most important one in the variation of the amount of methane found.

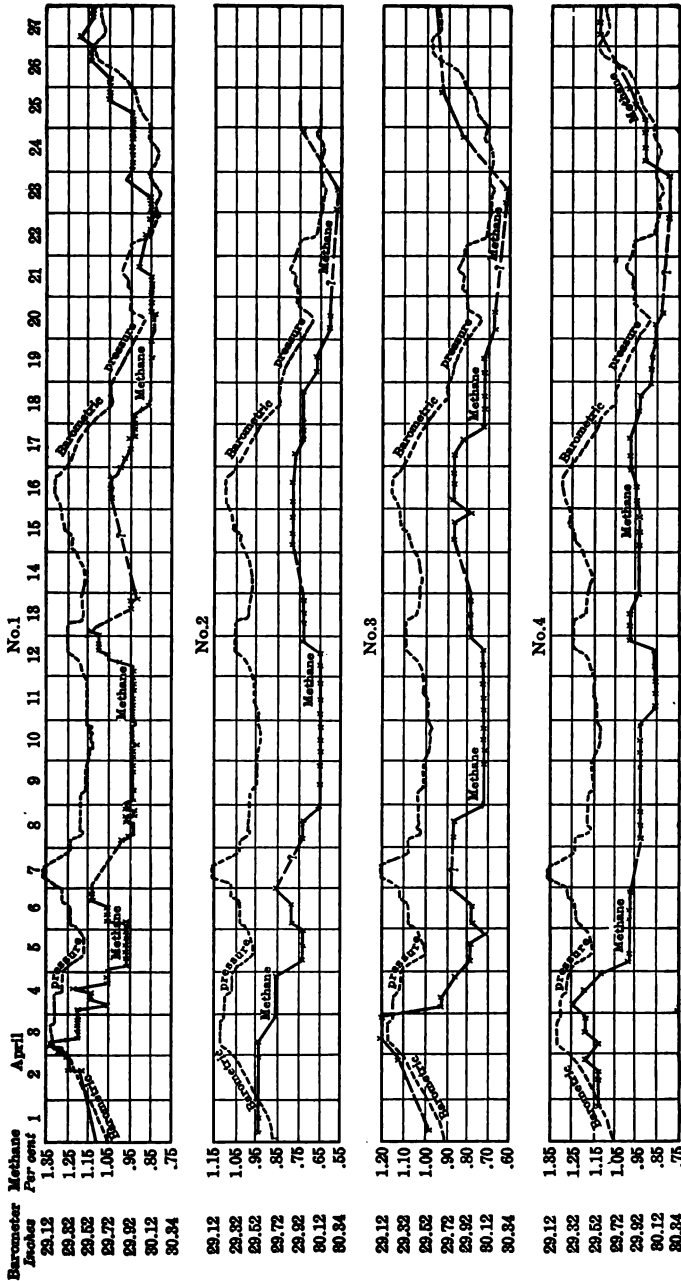


FIGURE 14.—Curves showing variations in percentage of methane and barometric pressure in Du Souich workings, Liévin, France, during April, 1907. No. 1, return air at 345-meter level, 16,800 cubic feet a minute. No. 2, south return air at 345-meter level, 14,076 cubic feet per minute. No. 3, east return air at 345-meter level, 7,653 cubic feet per minute. No. 4, return air in No. 4 gangway at 430-meter level, 4,748 cubic feet per minute.

In order to test this question further Morin made some precise determinations of the amount of methane in old workings at Liévin.

In a chamber in the most gaseous part of the mine, a tube 16 feet long was fastened along the roof and the chamber was then filled with gob. The end of the tube that projected into the gangway was plugged. After three months, samples drawn from the pipe contained as much as 0.46 per cent methane. A similar experiment in another chamber gave 0.75 per cent. Meanwhile the amount of methane in the ventilating currents in returns near by varied from 0.3 to 0.6 per cent. In old workings in another part of the mine where the ventilation was mostly cut off, some of the air from the gob was found to contain 2 to 2.7 per cent of methane. These tests indicate that although some gas is given off by old workings, even if closely filled, it does not increase to any important extent the volume in the returns. No air rich in gas could be obtained from any part of the old workings at Liévin even though sampling pipes were placed close to the roof and in the highest places.

At one stage of Morin's observations the ventilation of the mine was changed, with the result that when the volume of air passing through one of the main returns was diminished, the proportion of methane became only 0.4 per cent. Samples of this air taken at intervals, therefore, did not show any concordance with change of atmospheric pressure, although the air traversed a considerable area of old workings. In some working faces considerable gas came from inclosing strata, but the amount was variable and a much greater volume came from the coal in the face and from piles of broken coal.

RELATION OF BAROMETRIC PRESSURE TO AMOUNT OF GAS FROM FRESH COAL SURFACES.

In order to determine whether there was any relation between the barometric pressure and the amount of methane given off by fresh coal surfaces Morin made a series of tests of the return air from an individual chamber. The place selected was in the deep-lying Leonard bed and the return air from it was sampled every hour for two days, with the following results:

Percentages of methane at various atmospheric pressures in chamber in Liévin mine.

Barometric pressure of mercury.	Volume of methane per minute.	Barometric pressure of mercury.	Volume of methane per minute.	Barometric pressure of mercury.	Volume of methane per minute.
<i>Inches.</i>	<i>Cubic feet.</i>	<i>Inches.</i>	<i>Cubic feet.</i>	<i>Inches.</i>	<i>Cubic feet.</i>
29.72	17	29.90	18½	30.16	12
29.74	12	29.94	20	30.16	17
29.76	16	29.98	26	30.18	18
29.78	24	30.02	26	30.20	22
29.80	21	30.04	23½	30.24	34
29.84	27	30.08	26	30.26	33
29.84	18	30.08	26	30.28	38
29.86	15	30.10	26	30.30	31
29.86	13	30.12	22	30.32	41

This series of observations shows no relation whatever between the atmospheric pressure and the volume of methane given off, but the gas may have been coming out of the coal at this place at such pressure that the volume was not affected by the gradual $\frac{1}{2}$ -inch rise of the barometric pressure.

EFFECT OF SETTLING OF ROOF AND FLOOR.

The negative results of this test, confirming the supposition that the high internal pressure of the gas in the solid coal would not be affected by barometric change, and the lack of evidence that any great amount of gas came out of the gob and old workings, led Morin to conclude that the variations in the rate of escape of the gas were due to other causes. He suggests, therefore, that the barometric changes must mainly affect the volume of gas coming from floor and roof, whether this gas is contained in the rocks themselves or comes from

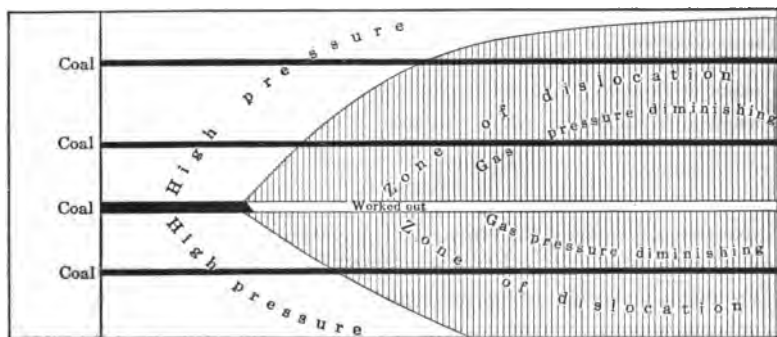


FIGURE 15.—Ideal section through coal workings showing area in which flow of gas is affected by removal of coal.

fissures connected with adjoining coal beds. He points out that as the coal is removed the roof settles and the floor tends to rise, causing fractures from which gas from adjoining virgin coal may enter the workings. The pressure affecting the flow of gas through such fractures is independent of the original pressure in the relatively impermeable coal and may be so light as to be closely balanced by atmospheric pressure. Morin believes that this source of gas is the most susceptible to atmospheric changes and is responsible for the great variations in volume of methane which are illustrated in figure 15.

RELATIVE VOLUMES FROM DIFFERENT WORKINGS UNDER VARIOUS CONDITIONS.

In order to ascertain the relative amounts of methane given off by workings of various kinds and extents under changing atmospheric pressures Morin collected samples of return air from various splits of return No. 1 at Liévin before and after a barometric fall from 30.72 to 29.12 inches, December 6 to 14, 1907. The results were as follows:

Volumes of methane in returns to shaft No. 1 at Liévin, Dec. 6 to 14, 1907.

Return.	Surface uncovered.	Voids in old workings.	Volume of methane per minute. ^a		Remarks.
			Dec. 6, barometer, 30.12 inches.	Dec. 14, barometer, 29.12 inches.	
	<i>Sq. yards.</i>	<i>Cu. yards.</i>	<i>Cubic feet.</i> 247	<i>Cubic feet.</i> 247	
Main return to shaft No. 1.			13	13	
Arago, at 534-meter level.....					Gangway 1,300 feet long with blower.
Auguste, N return.....	3,600	1,703	4	4	
Frederic, H return.....	27,600	14,410	13½	19½	Small workings with rise of 800 feet.
Du Souich at 476-meter level.	18,000	15,720	20	26	
Auguste, 31 return.....	31,200	19,650	7	7	Overlying and underlying bed worked out.
Edouard couchant return F..	25,200	20,960	18½	21	
Frederic, return 200.....	40,800	21,339	16	37	
Auguste and Du Souich return 0.	43,800	24,628	19	27	
Du Souich return 200.....	42,000	30,130	16	24	
Alfred return at 345-meter level.	90,000	52,400	19	24	
East return at 345-meter level.	109,200	68,120	22½	49	
Alfred at 476-meter level....	114,000	78,000	21½	30	
East return at 283-meter level.	166,800	103,490	28	53	
South return at 345-meter level.	180,000	144,100	109	128	
South return at 283-meter level.	391,200	262,000	171	223	

^a Approximate.

These results show that volume of gas in old workings is not the sole factor to consider. In all the workings but those on Auguste return No. 31 the adjoining beds had not been mined. Old workings in beds of coal equally gaseous and worked to the same extent gave off greatly different amounts of gas from the same volume of voids under the 1.6 inch of atmospheric variation shown in the table. The volume of gas in airway 31 from workings No. 1 in the Auguste bed and from workings in adjoining beds did not vary during the great fall of pressure. On the other hand, in airway 200 the methane from workings in the Frederic bed, which are undermined in about the same area by workings in the Du Souich bed, 50 feet below, was more than double that from the Du Souich workings; also the volume of gas from the latter workings varied less with the change of atmospheric pressure, indicating an escape of gas upward from the Du Souich bed to the Frederic bed workings.

The volume of gas in Auguste airway N, which drains a small area of workings and a rise 800 feet long, did not vary; neither did the volume of gas in the air from the Arago bed, in which a gangway 1,300 feet long was in progress and over which there were many blowers. In this case the emission of gas from coal in place and from blowers was not affected by a great change of atmospheric pressure.

CONCLUSIONS DERIVED FROM RESULTS OF TESTS AT LIÉVIN.

The tests show that the virgin coal is only slightly permeable and therefore the gas pressure in the rock strata affected by the mining is independent of the pressure in the virgin coal, so that as the gas is drained by advance of the workings the pressure becomes feeble. From the facts observed the following conclusions are drawn:

1. The pressure and volume of gas in the rock strata (near the working face) is low.
2. The coal beds are less gaseous near the working faces than in more remote parts and in certain cases the drainage of the beds is complete and the last of the gas escapes under feeble pressure.
3. The coal beds far from workings remain gaseous.
4. The strata affected by coal workings, even though distant, contain little gas, and in consequence after a settling of roof or rise of floor the gas pressure in the zone of fracturing remains very feeble.

Morin gives essentially the following résumé of the experiments in the Du Souich bed:

The quantity of gas liberated does not depend alone on the amount of change of barometric pressure, but also on the absolute pressure, and therefore becomes proportional to the time between the two extremes of pressure limiting the fluctuation. When a period of constant atmospheric pressure followed a fluctuation the proportion of gas remained at the figure that it had reached as a result of the fluctuation (see curves in No. 1, fig. 14). A low or a high pressure lasting for several hours induced a maximum or a minimum proportion of gas lasting for the same length of time. A given pressure was not always attended by the same proportion of gas because of complications caused by the effects on the atmosphere in old workings. After an increase to a given atmospheric pressure the percentage of gas was smaller than after a diminution to the same pressure. He suggested that probably this could be explained by the supposition that during a period of high pressure the gas accumulates in old workings, whence during periods of low pressure the gas flows out into the returns.

Finally Morin concludes that variations in gas outflow induced by fluctuations in atmospheric pressure are due not only to old workings but also to gas from adjoining strata (roof and floor) and from fissures through these strata which admit gas from adjoining coal seams.

OBSERVATIONS IN ENGLAND AND THE UNITED STATES.

At a meeting of the Midland Institute of Mining Engineers one of the mine managers ^a stated that when the mine was hot—that is, had relatively high temperature—the presence of methane was less evident,

^a Anon. The barometer as an indicator of gas in collieries: Min. and Sci. Press, vol. 34, 1877, p. 10.

but when an increased volume of cold air was introduced the proportion of gas increased rapidly. This had been noted many times. Therefore, the variations of temperature must have had a close relation to the amount of gas given off in this mine.

For several years at the Auchincloss mine, near Nanticoke, Pa., a comparison was made of barometer readings and daily reports of fire bosses as to the amount of gas in the headings, and it was found that there was no relation between them. The mine is relatively new, without large areas of old workings.

TESTS WITH SAMPLES OF PENNSYLVANIA COAL.

In tests to determine the amounts of gas given off under diminished pressure Chamberlin ^a obtained some results that throw light on the effect of barometric changes. The relative amounts of gas given off by bituminous and anthracite coals under long-continued low pressure were compared with those obtained at ordinary air pressure. Two samples were taken, one of bituminous coal from the Mansfield mine, Carnegie, Pa., and one of anthracite coal from Nanticoke, Pa. Each sample was divided into two parts, one of which was placed in a vacuum bottle from which the air was pumped; the other was sealed in a bottle at ordinary air pressure. All the bottles were submerged in water. After seven days the anthracite samples were tested and in both bottles the methane evolved was 1.07 times the volume of the coal. At the end of 10 days the bituminous coal kept in a vacuum had given up 0.25 of its volume of methane, whereas the sample under ordinary air pressure had given off only 0.14 times its volume. As the amount of methane evolved in the air-filled bottles was greater than the volume of oxygen absorbed from the air by the coal, the final pressure was slightly greater than the barometric pressure toward the close of the test.

In these experiments the diminished pressure appeared not to have increased the emanation of gas from the anthracite. Chamberlin decided that barometric variations that do not exceed $1\frac{1}{2}$ inches can not have much effect on gas escaping from the interstices of the coal, and that time was the principal factor in the escape of gas. Blowers and feeders escaping under low pressure from fissures are naturally much more sensitive and might be affected by barometric changes.

EFFECTS OF EARTH MOVEMENTS ON METHANE EMANATION.

Many suggestions have been made as to the effects of earthquakes, sun spots, and other natural phenomena on the emanation of gas in coal mines, but most of the evidence presented is not convincing.

^a Chamberlin, R. T., Notes on explosive mine gases and dusts, with especial reference to explosions in the Monongah, Darr, and Naomi coal mines: Bull. 26, Bureau of Mines, pp. 37-39; reprint of U. S. Geol. Survey Bull. 383.

Several papers on the subject give comparative statistics of explosions, overlooking the fact that many of them are caused by coal dust and due to some mishap in the mine that would produce the same result at any time. The suggestion that variations in atmospheric pressure cause an up-and-down movement of the earth's crust, which changes the conditions of gas emission, may have some bearing. Darwin ^a estimates that very high barometric pressure—good weather conditions—may bend down the earth's crust 3 or 4 inches, and conversely diminished pressure may be followed by expansion. The latter might cause the opening of crevices and let out gas, especially where mining has set up uneven stresses in the strata. Gas being under great pressure in the strata its emission might be influenced by a cause of this kind.

Milne ^b has presented an extended discussion of the effect of the earth pulsations due to meteorological influences. He cites experiments of Chesneau ^c at Anzin, in France, who for many months noted relative movements of a tromometer and amount of methane given off. The relation was well marked in some cases but in general it kept pace with barometric changes.

In figure 16 are shown results of some of Chesneau's observations made for 11 months in the gaseous workings of the Herin mine near Anzin. The curves show the percentage of methane, barograph readings, and the tromometer record of earth tremors. The latter were mostly very slight, except the notable pulsation on December 8, which was at a time of rapidly diminishing atmospheric pressure and attended by great increase in methane content in the return air. The details from December 6 to 10 are shown on a large scale in figure 17.

Articles advocating the belief in a relation between seismic disturbance and mine explosions have been published by Beard,^d Spalding,^e Stow,^f and others.

From a consideration of the evidence presented it appears to the author that earth movements can not affect the emanation of methane unless possibly to precipitate an outburst that is nearly ready to take place; ordinarily a blast in a near-by chamber would cause much more local movement.

^a Darwin, G. H., On mechanical effects of barometric pressure on the earth's surface: Rept. British Asso. Adv. Sci., 1882, pp. 106-111.

^b Milne, J., On earth pulsations and mine gas: Trans. Fed. Inst. Min. Eng., vol. 5, 1893, pp. 203-230.

^c Chesneau, G., and others, De l'étude des mouvements de l'écorce terrestre poursuivie particulièrement au point de vue de leur rapports avec les dégagements de produits gazeux: Annales des mines, ser. 8, vol. 9, 1886, pp. 207-281; and

Chesneau, G., De l'influence des mouvements du sol et des variations de la pression atmosphérique sur les dégagements de grisou: Annales des mines, ser. 8, vol. 13, pp. 389-428.

^d Beard, J. T., Colliery explosions and their causes * * * in which seismic forces are believed to be more important than barometric changes: Eng. and Min. Jour., vol. 83, 1907, pp. 1051-1054.

^e Spalding, W. A., Mine explosions as related to earthquakes: Eng. and Min. Jour., vol. 87, 1909, pp. 411-413.

^f Stow, A. H., Seismic disturbances and coal-mine explosions: Eng. and Min. Jour., vol. 88, 1909, pp. 449-450.

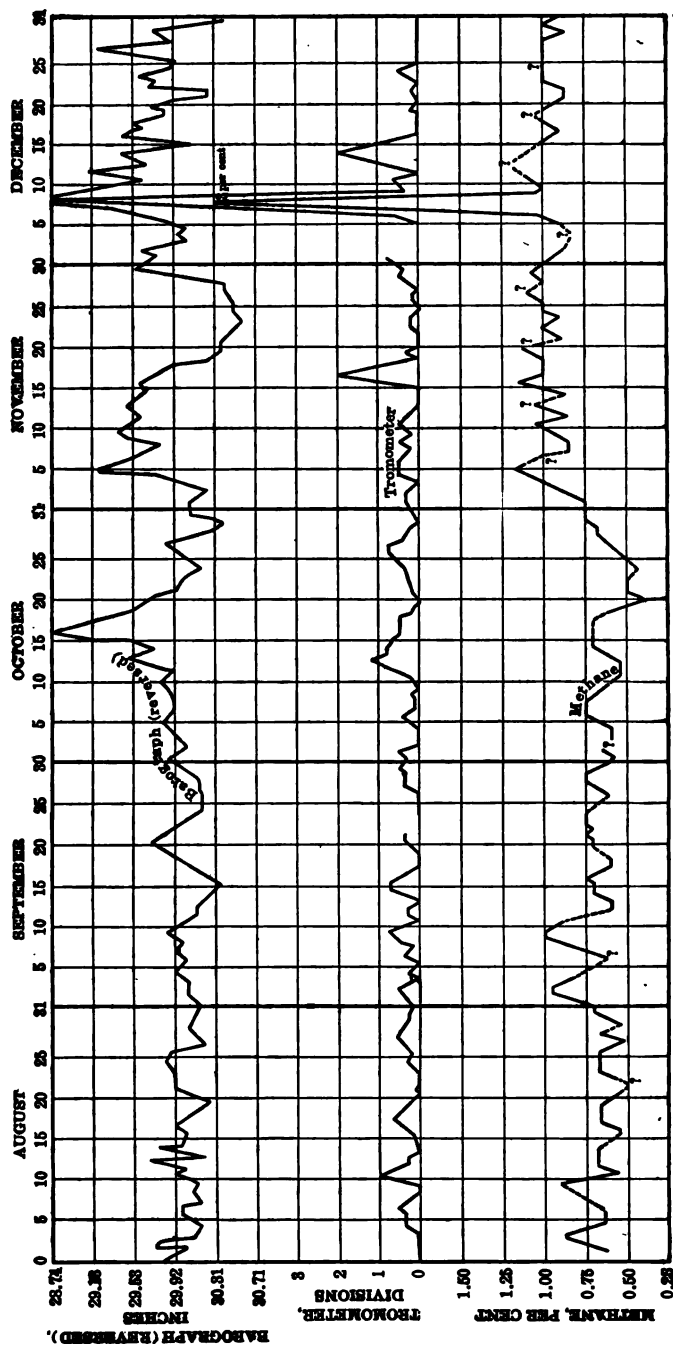


FIGURE 16.—Curves showing variations in atmospheric pressure, percentage of methane in return air, and earth tremors at the Herin mine, Anzin, France, August to December, 1896.

DURATION OF METHANE EMANATION FROM COAL.

Many of the facts stated on previous pages show that coal loses the greater part of its gas content while it is being mined and broken, and in the same way much of the gas in the coal exposed in faces, ribs, and pillars drains off rapidly. In the mine, however, this drainage of gas is superficial, for at greater or less depth the solid coal contains its ordinary volume. This gas is disposed to flow to the exposed surface, but the rate of movement must depend on the pressure and volume of the gas and the porosity and solidity of the coal. In general, in gaseous places it is found that the gas emanation is rapid when the workings are progressing rapidly, but after awhile as the district is opened, the volume of gas greatly diminishes. It is a frequent practice in gaseous mines to extend gangways rapidly into gaseous places to effect a general ventilation and then to leave the place for awhile until the gas diminishes so that mining ceases to be dangerous. Slight cracking of roof and floor facilitate gas escape and in some cases it gives rise to feeders. The diminution of methane emanation is shown by the fact that in many mines after Sunday cessation or any other shutdown the amount of gas diminishes. There are many examples of gaseous beds that have yielded little gas in areas where the adjoining beds have been worked out, the gas having drained off through cracks and porous strata.

GAS DRAINAGE IN MINING.

The project of tapping off gas in gaseous districts is one that many coal engineers have considered and occasionally have carried into practice. In many mines it has been found desirable to extend gangways rapidly through a gaseous district and then after ventilation has been established to let the gas draw off for weeks or even years before general mining is begun. Working out an overlying bed in order to drain off gas from the lower one is often tried, but the efficacy of drainage depends on the permeability of the intervening strata. The extension of bore holes horizontally in the coal bed or from the surface above is a promising expedient, but in most cases it does not draw much gas from solid coal because the surface exposed is too small and gas travels through the coal too slowly to affect a

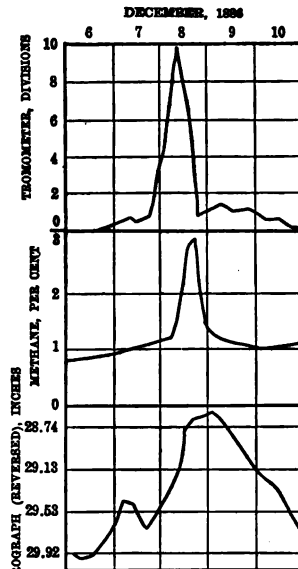


FIGURE 17.—Curves showing variations in atmospheric pressure, percentage of methane in return air, and earth tremors at Herin mine, Anzin, France, December 6 to 10, 1896.

wide area. This has been proved by tests of pressure and volume from bore holes; after a hole has been opened a long time and then closed the pressure is soon the same as the original. Many tests have been made of drainage holes in European mines, but the results have been entirely unsatisfactory. Some experiments by Lange^a in Rochebelle mines, France, to drain off carbon dioxide by bore holes were unsuccessful. Old workings, however, have been drained by tubes sunk from the surface, sometimes to great advantage. The author observed an instance of this in the Otto colliery, 5 miles west of Pottsville, Pa., where tubes tapped the top of workings in inclined beds and a very large volume of methane was brought up by escaping air from a small ventilation split. The proportion of gas was so high that the mixture could be lighted and would burn for hours.

RELATION OF METHANE CONTENT TO CHARACTER OF COAL.

The investigations having shown that the amount of methane in coal varies greatly in different coal beds in the same series and in different parts of the same bed, we conclude that the original methane content is closely related to the constituents of the original deposit, possibly modified somewhat by conditions under which the coal was formed. In making comparisons, however, the original amount of methane should be taken into consideration, because many parts of the coal bed that may have been very gaseous when they were formed have lost more or less of the gas by drainage to the surface. Such drainage through overlying strata takes place partly by percolation and in part by crevices due to fracture. It is also a fact that the gas emitted in outbursts, blowers, and even in notably gaseous workings is not always an indication of the total amount of methane in the coal that may be given off in time. No relation has yet been found between the proportion of methane and the texture of the coal or of the components as indicated by chemical analyses, for coals high in volatile constituents are not the highest in methane. Undoubtedly the original vegetal material now represented by coal varied greatly in character in different places and at different times even in the same basin, and the rate of decomposition and of burial by later deposits and the nature of these deposits varied greatly and these variations had much to do with the development of gases. It is the author's belief that a relation will be found between methane content, or at least the original methane content, and the organic constituents of the coal, and it is along this line that future detailed study of the problem should be directed.

^a Lange, C., Les dégagements d'acide carbonique aux mines de Rochebelle: Bull. Soc. l'indus. min., ser. 3, vol. 6, 1892, pp. 1141-1180.

**PROBABLE ERROR IN FIGURES FOR METHANE CONTENT
IN MINE AIR.**

In most records of the proportion of methane in mine air the figures are given to the second place of decimals and even with greater apparent precision. In an investigation of the probable accuracy of ordinary methane determination G. A. Burrell,^a chemist of the Bureau of Mines, has found that the probable error is nearly 0.03 per cent. Accordingly, in two analyses reported as 0.17 and 0.19 per cent, for example, the amount of methane may be the same, or the first may be 0.19 per cent and the second 0.16 per cent. This probable error invalidates all arguments based on such small differences as 0.03 per cent. When this probable error is calculated into cubic feet per minute the error increases, for air volume determinations are necessarily somewhat inaccurate. Usually the area of cross section is not precisely determinable or determined. Most anemometers have an error of at least 1 per cent; in many of them the amount is much greater, and moreover the actual velocity of the air current varies in different parts of the cross section. Probably the ordinary determination of volume of air per minute has an error of at least 5 per cent. That is, a current reported as 10,000 cubic feet a minute may be anywhere between 10,250 and 9,750 cubic feet. Therefore, in such a volume of air analyzing 1.0 per cent of methane the true amount of that gas is between $102\frac{1}{4}$ and $97\frac{1}{4}$ cubic feet per minute. This is not allowing for probable error of analysis, by which the true proportion may range from 1.01 to 0.98 per cent, so that the true volume of methane in this instance is between 103.5 and 95.3 cubic feet a minute, a possible difference of 8.2 cubic feet, or nearly 0.1 per cent, if the errors are all negative. It is the author's belief that it would be safe to assume a possible error of 0.1 per cent in all the reports of volume of 50 cubic feet or more and a somewhat larger error for smaller amounts, increasing rapidly to 50 per cent when the reported methane percentage is near 0.03 per cent. Velocity movements are also less reliable when very low or very high, especially where the cross section is large and irregular.

^a In a letter to the author.

EXPLOSIVE GAS IN THE NORTHERN ANTHRACITE COAL FIELD OF PENNSYLVANIA.

INTRODUCTION.

In beginning the investigation of the conditions under which explosive gases occur in coal beds in the United States, the anthracite coal field of Pennsylvania was selected on account of the structural conditions presented there. As one of the objects of the investigation was to determine whether there was any relation between methane emanation and the structure of the coal measures, the many flexures of the anthracite beds and the great extent of the mine workings in them led to the selection of such beds for study. A general examination of the entire anthracite region led to the conclusion that the most instructive results would be obtained by a detailed study of certain typical gaseous mines in the northern basin near Wilkes-Barre and Scranton. The results are presented in the following pages.

Some of these mines produce large volumes of methane, several giving off 2,000 to 3,000 cubic feet a minute. The gas is brought to the surface by the ventilating fans and constitutes 1 to 2 per cent of the return air. The volume of methane issuing from three mines at Wilkes-Barre is 9,000,000 cubic feet a day, a volume equal to that of the illuminating gas from a plant supplying a city of 300,000 inhabitants.

In making the investigation attention was given to the effect of depth, of the nature of the beds overlying the coal, and of the area of coal surface exposed, on the occurrence of gas in the mines. Many samples of return air were taken and sent to the Pittsburgh laboratory of the Bureau of Mines, where they were analyzed. These samples were collected in December, 1910, and January, 1911, and some supplemental sampling was done for the author by H. I. Smith, assistant mining engineer of the bureau, in February, March, and April, 1912. Later, in April, 1912, after mining had been suspended nearly a month, the author again visited certain mines and resampled the air to obtain data for comparison with previous samples taken when the mines were in operation. By this means it was possible to ascertain the influence of a cessation of mining on the escape of methane.

Anthracite is fully as gaseous as bituminous coal, and so far as can be seen the gas occurs under the same conditions in both classes of coal. In a test made of a sample of anthracite from a mine at Nan-

ticoke, Pa., the gas given off was more than three times the volume of the coal and some had been lost in mining, etc. In the northern anthracite basin small gas explosions occur nearly every day, and although many of them do no great damage, yet men are fatally burned in some of them. In the 11 years from 1899 to 1909, the number of deaths from explosions of coal gas in the anthracite mines was 245; of these about four-fifths were in Luzerne County. In each of these explosions the body of gas was small and only a few persons were killed, some by burning and others by afterdamp. The mine operators have to use great precaution to insure complete ventilation in all working chambers of the mines, and in some districts a large volume of air has to be forced through the mine to dilute the gas below the danger point. According to the report of the department of mines of Pennsylvania, the volume of air discharged from all the mines in the northern basin is nearly 31,000,000 cubic feet a minute, most of it being drawn by fans driven by steam or electric power.

As the average daily output of coal from the basin is about 131,000 tons, the volume of air for ventilation is equal to about 341,000 cubic feet for every ton of coal mined.

The number of mines is about 200 and the total number of men underground nearly 64,000. The State law requires 200 cubic feet of air per person in the mine, but in nearly all mines this amount is greatly exceeded and in the gaseous district about Wilkes-Barre three to four times this amount is used. The gross volumes in each district, as measured in upcasts, is given in the following table, which shows also the number of miners. The amount of methane in these returns varies from an almost imperceptible trace to 2 per cent. It is not possible to estimate the average proportion of methane, but in the proportion of 0.1 per cent, which is not excessive, there would be in the returns from all these mines about 31,000 cubic feet of methane a minute or 5,760,000 cubic feet a day, weighing 130 tons.

Volume of air and number of miners in the mines in the northern anthracite coal basin.^a

District.	Air (upcast) per minute.	Number of miners.
	<i>Cubic feet.</i>	
1. Coal Brook to Sterrick Creek mines	2,433,250	6,232
2. Olyphant to Legett Creek mines	2,586,703	6,748
3. Scranton region	3,080,691	6,045
4. Bellevue to Minooka mines	2,433,616	6,240
5. Old Forge to Austin mines	2,002,708	5,322
6. Barnum mine to Reliance mine	3,085,334	7,590
7. Wilkes-Barre region	5,129,532	6,832
8. Exeter to Kingston No. 4 mine	3,371,389	6,140
9. Kingston colliery to Chauncey mine	3,791,800	7,796
10. Susquehanna Coal Co. 5-7 Wanamie-West End mines	2,991,182	5,043
Total	30,906,705	63,988

^a Compiled from report of Department of Mines of Pennsylvania for 1909.

GENERAL STRUCTURE OF THE NORTHERN ANTHRACITE BASIN.

The northern anthracite field in Pennsylvania is a narrow canoe-shaped basin, 55 miles long on a general northeast course, 6 miles wide at its widest part and tapering to a point at either end. The coal measures are in a syncline that is flat-bottomed and regular to the northeast but more closely and deeply flexed and crenulated to the southwest. The thick-

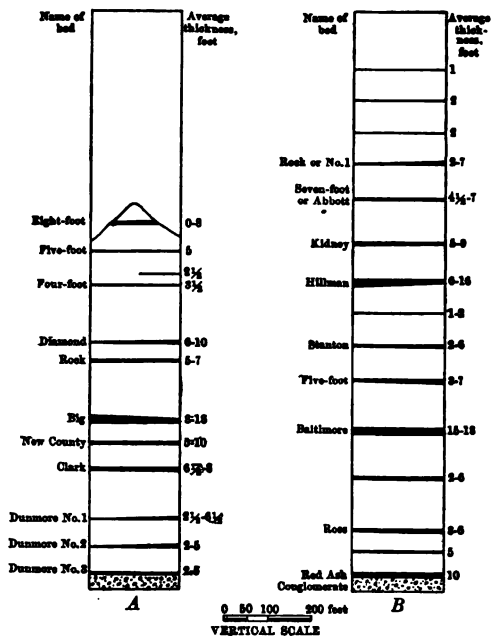


FIGURE 18.—Columnar sections showing stratigraphic position of coal beds in northern anthracite basin. *A*, near Scranton; *B*, near Wilkes-Barre.

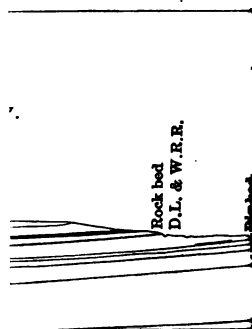
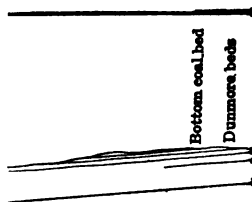
ness of the coal measures in this syncline is nearly 1,000 feet in the northern part about Scranton, and more than 2,200 feet in the deep basins southwest of Wilkes-Barre. The rocks are largely sandstone and "slate," and about 10 beds of coal now regarded as workable occur at irregular intervals in the measures. The general succession in two parts of the field is shown in figure 18.

These beds have all been mined and in places some of them have been removed under areas of considerable extent. It has been estimated that the total original amount of coal was 5,666,219,784 tons and that up to the end of 1912 slightly

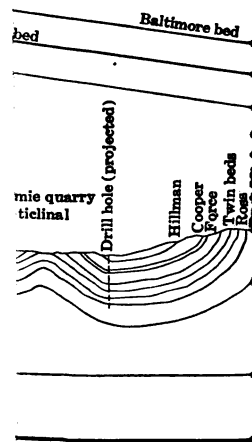
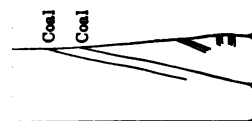
more than 1,000,000,000 tons had been removed. The present production is approximately 50,000,000 tons a year.

The coal beds lie at various depths, according to their position in the measures and the geologic structure; they all outcrop at either end of the basin and along the sides. In a wide area in the Scranton region the lower coal beds lie between sea level and 300 feet above sea level or 500 to 700 feet deep, but near Pittston and Wilkes-Barre where there are deep flexures they are 1,000 to 1,500 feet below sea level, or 1,500 to 2,200 feet below the surface. The general structure of the basin at intervals along its length is shown in the cross sections in Plate I and by the contour lines in Plate II.

The contour lines in Plate II were traced from a map showing the structure of the basin in detail, which is to be published later. In



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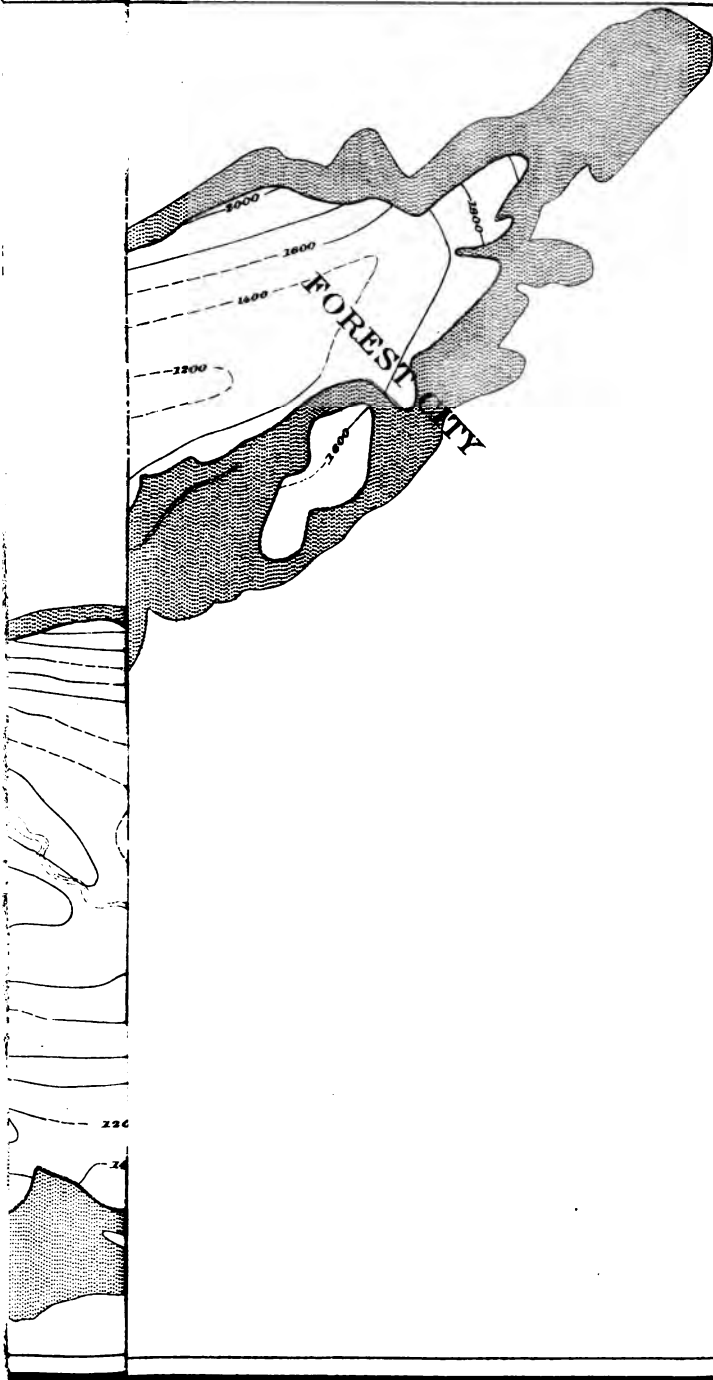


COITE COAL BASIN.

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WILLIAM L. BROWN





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order to determine the relations between the occurrence of gas and the structural features, a complete compilation was made of the position of the coal beds as shown by mine maps, especially of workings in the lower beds and further data were obtained from many bore-hole records kindly made available by officials of the various companies. From this material contour lines were drawn to show the configuration of the lowest important coal bed, which is called the Red Ash in the Pittston-Shickshinny region and the lower Dunmore in the Scranton region. In certain areas in which no mining had been done, and in places where only the upper beds have been mined, the structure had to be inferred by various means and the hypothetical contours are represented by broken lines.

COLLECTION OF MINE-AIR SAMPLES.^a

Samples of mine air were collected in bottles which were taken down in the mine and filled with mine air by pumping out the contained air with a rubber-bulb aspirator. In some of the work half-pint bottles with glass stoppers were used, the stoppers being slightly greased to insure a close fit. In many cases the stoppers were either sealed by dipping in paraffin or tied by a loop of cord. They were placed in boxes and so wedged by soft paper packing that there was no chance for the stoppers to shake loose. In other mines bottles such as are made for holding citrate of magnesia were used; these had the usual stopper of porcelain and rubber washer held down by the latch of the wire loop. They were sealed with paraffin. A few tin bottles with tubular ends closed by rubber tubes and glass plugs were used, but these were liable to leakage unless handled with special care. Bottles closed with sound corks boiled in paraffin and sealed with paraffin were found satisfactory when citrate of magnesia bottles were not obtainable. The rubber-bulb aspirator was an ordinary bulb with valve, such as are used in syringes, and a 1-foot length of small tube. The bulb was squeezed about 70 times to exhaust the original air from the bottles. It is well to test such an aspirator frequently for leakage, by ascertaining whether the squeezed bulb remains collapsed when the inlet is held closed and resists compression when full if the outlet is closed. The air may also be sucked out of the bottle by means of a rubber or other tube, by 20 or more inhalations. The bottles may be filled with water and emptied at the place of sampling, but as carbon dioxide is absorbed by water, there may be a slight loss of that gas from confining the sample in a wet bottle and from washing some of the intruding air with the outflowing water.

^a For detailed discussion of the subject see Burrell, G. A., and Siebert, F. M.: The sampling and examination of mine gases and natural gas: Bull. 42, Bureau of Mines, 1913, 116 pp., 2 pls., 23 figs.

In collecting an air sample in the mines care was taken to move the bottle about near the top, bottom, and each side of a chamber, thus insuring a fair average. Where there is a strong current the air and gases are well mixed, but in places where returns join or there are local emanations of gas a carelessly taken sample may not be representative of the whole volume of air. Measurements of air currents, made at the time of collecting samples, gave the average movement of the air current in the entire cross section. They were repeated so as to eliminate local variations caused by opening and shutting doors, blocking of the shaft by cages, movement of trips of cars along gangways, and other conditions that tend to affect the air velocity.

CONDITIONS IN REPRESENTATIVE MINES.

LANCE MINE.

The Lance colliery is in the upper part of Plymouth. Its workings are largely on the north slope of the main basin, but some of them extend south across the broad syncline under the Susquehanna River. In the latter area all the principal coal beds of the region are represented, but as the north pitch is ascended some of the higher beds come to the surface. The mine is especially gaseous, the gas being derived from various beds. Although some parts of the mine are much more gaseous than others and numerous large feeders have been encountered, considerable gas is given off throughout the deeper workings.

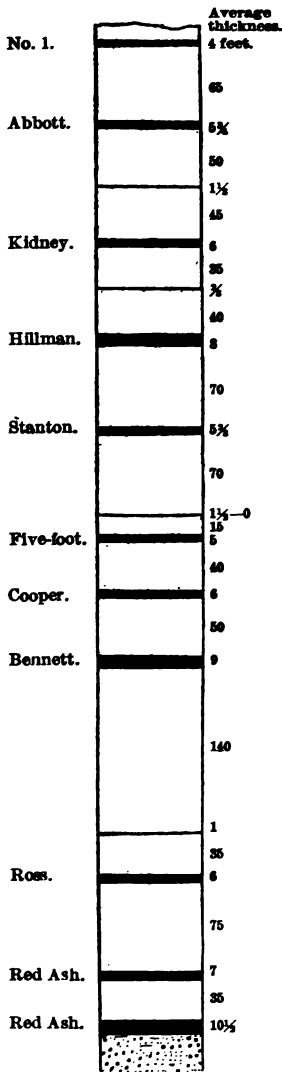


FIGURE 19.—Columnar section of coal measures at Lance mine, Plymouth, Pa.

SUCCESION AND STRUCTURE OF BEDS.

Under the greater part of the Lance mine about the shafts and southward under the river there are 11 thick beds of coal which occur in

the succession shown in figures 19. The thicknesses given are averages and there is considerable local variation. The coal beds aggregate 75 feet in thickness but only about 61 feet are coal, nearly every one of the thicker beds carrying 6 inches or more of "slate" or bone, the Bennett and Stanton beds having nearly 2 feet in parts of their areas. The general structure of the Lance mine, from the north on the mountain slope to the southern margin of the mine on the south side of the basin, is shown in figure 20. This section shows the thicker coal beds slightly exaggerated in thickness and also the thick body of river deposits covering the measures in the river flat south of the shaft. These deposits are more than 300 feet thick in places.^a

RETURN-AIR SAMPLES.

As the Lance mine presented varied conditions of structure, gas occurrence, depth, and surface cover, it was selected for an extended investigation. On January 7 and 9, 1911, samples of return air were taken at various points where they would be representative of certain coal beds and working areas. A year later some additional samples were taken by H. I. Smith, and the author also resampled several returns after mining had been discontinued a month. These samples were analyzed in the laboratories of the bureau at Pittsburgh, Pa., with the results given in the table following.

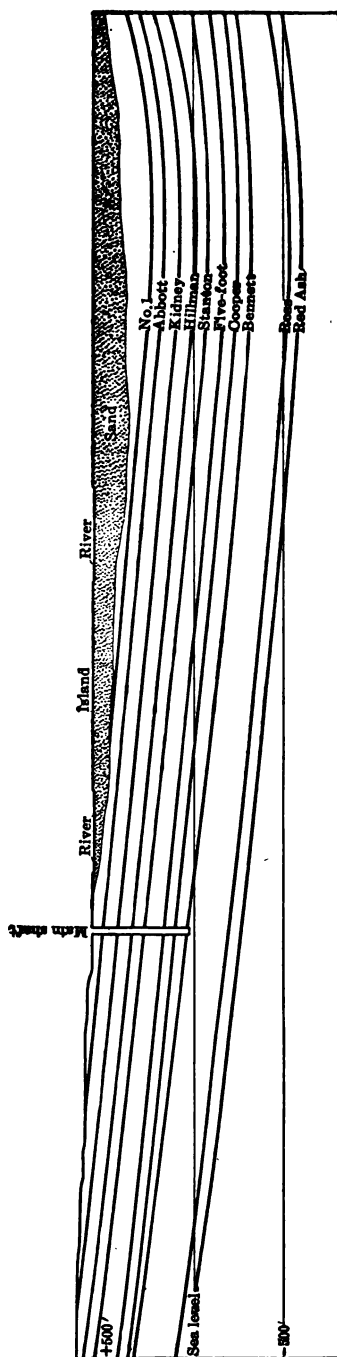


FIGURE 20.—Generalized cross section of Lance mine, Plymouth, Pa.

^a See Darton, N. H., Sand available for filling mine workings in the northern anthracite basin in Pennsylvania: Bull. 45, Bureau of Mines, 1913, 33 pp., 8 pls., 5 figs.

Results of analyses of return air in Lance mine.

[G. A. Burrell, analyst.]

Laboratory No.	Return.	Place of sampling.	Velocity of air current per minute.	Volume of air current per minute.	Proportion of methane in return air.	Volume of methane per minute.	Proportion of CO ₂ in return air.
			<i>Cubic ft.</i>	<i>Cubic ft.</i>	<i>Per cent.</i>	<i>Cubic ft.</i>	<i>Per cent.</i>
1267	Bennett and overlying beds.	Total upcast, No. 1 air shaft.		200,000	0.96	1,920±	0.10
1271	Red ash, total.	Near No. 2 air shaft....	950	99,750	1.07	1,067	.17
2192	Bottom Red Ash.	Just south of No. 2 air shaft. ^a	721	75,000	.99	742	.12
2197	Red Ash.....	400 feet from 4th east gangway. ^a	309	22,900	.65	149	.11
2195	Red Ash?.....	Overcast over 6th slope below 4th east gangway. ^a	347	30,700	1.59	276	.17
2209	Ross.....	At No. 2 air shaft ^a	593	40,900	1.13	462	.06
1270	Ross, total.	No. 5 slope.....	873	48,888	1.46	714	.04
1265	Bennett.....	No. 25 tunnel.....	486	21,663	1.32	70	.07
1275	Do.....	No. 2 slope.....	473	41,505	1.97	818	.05
1264	Do.....	No. 4 slope near big feeder.	377	15,080	2.39	360	.10
2171	Do.....	Bottom of No. 1 air shaft ^a	452	49,300	.57	281	.04
1277	Cooper, west.....	In No. 11 tunnel.....	267	18,957	2.68	508	.13
1272	Cooper, east.....	In No. 23 tunnel.....	364	41,500	1.04	431	.06
2172	Cooper.....	Near No. 1 air shaft....	420	44,150	1.20	529	.07
2198	Do.....	In No. 23 tunnel, 600 feet from No. 26 tunnel ^a	318	25,850	.91	217	.05
2201	Do.....	In No. 11 tunnel west, 1 mile from No. 1 slope. ^a	364+	33,124	2.04	676	.10
1269	Five-foot, east.....	At No. 7 slope.....	390	26,220	1.46	383	.08
2213	Do.....	do.....	185	13,000	.04	5	.05
1274	Five-foot, west.....	do.....	522	28,188	.23	65	.10
2211	Do.....	do.....	441	22,100	.16	35	.08
2191	Five-foot.....	Near No. 2 air shaft ^a ..	683	34,800	.93	324	.09
1276	Hillman, east.....	No. 14 tunnel.....	402	36,984	1.50	555	.10
1273	Hillman, west.....	do.....	723	30,366	.55	167	.10
2223	Hillman and Five-foot.	Total No. 7 slope at bottom of No. 1 air shaft on Five-foot bed. ^a	1,294	49,200	.73	350	.04

^a Collected by H. I. Smith, February, 1912.

That the Lance mine is a notable producer of methane is indicated by the fact that at No. 1 air shaft near the main hoisting shaft the methane content of the upcast air is 0.96 per cent or a volume of nearly 1,920 feet a minute, the volume of the air being about 200,000 cubic feet a minute. This air is from the Bennett, Cooper, Five-foot, Hillman, and overlying beds, from which the average daily production of coal at that time was 1,220 tons. Besides this upcast there was another fan at an air shaft 1,400 feet north which was bringing up from the Red Ash and Ross beds an additional 1,781 cubic feet of methane a minute at the time of the author's visit. The output of coal from these two beds was 710 tons a day.

RED ASH RETURNS.

The Red Ash workings in the Lance mine occupy the area shown in figure 21.

The workings north of the Susquehanna River are mostly old and most of the area being mined was under the river and islands, one

gangway extending south to the bottom of the basin. There is a regular south dip throughout the worked area, the coal bed descending from near sea level in the north end of the property to 600 feet below sea level, or 1,130 feet below the surface in the bottom of the basin. There are a few local structural irregularities of small extent and, as shown in figure 19, the bed splits into two parts. The two splits are separately mined in places, the top bed being about 6 to 7 feet thick and the bottom one 7 to 11 feet thick. To the northward, where they are thinner and less rock intervenes, coal from the two splits is taken out at the same time, and the thickness of the coal and intervening rock is 20 to 23 feet. The deeper Red Ash workings have developed considerable gas in places and a 10-inch bore hole to this bed from the workings in the Stanton bed yielded a flow of gas estimated at 1,000 feet a minute. At the time of sampling the return air (sample 1271) there were 51 working faces in operation and the average daily output of coal was 643 tons. The return air from these workings has its outlet through No. 2 air shaft, near which the volume was 99,750 feet a minute. The air contained 1.07 per cent methane, or 1,067 cubic feet a minute, and was high in carbon dioxide, 0.17 per cent. If the area of coal exposed in those workings be estimated at 1,463,000 square feet, 0.73 cubic foot of methane was liberated for every 1,000 square feet of coal exposed. With 643 tons daily production the rate was nearly 2,400 cubic feet of methane per ton of coal mined.

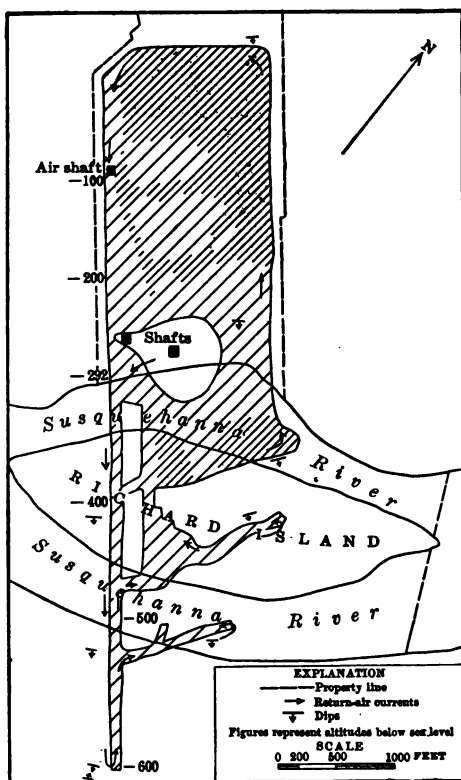


FIGURE 21.—Sketch map of worked area in Red Ash bed, Lance mine. The bed is mined in two splits in the area represented by double ruling.

ROSS RETURNS.

The Ross workings in the Lance mine, shown in figure 22, occupied very nearly the same area as the Red Ash workings but the slope to the southeast crosses the axis of the syncline and rises on the south

side. Only 12 places were being worked at the time of sampling and these were all in the extreme southern part of the mine. The average daily production was 66 tons. In most breasts $5\frac{1}{2}$ to 6 feet of coal was being mined. The air all went to No. 2 air shaft, which was passing 48,888 cubic feet of air a minute, the air containing 1.46 per cent methane, or 714 cubic feet a minute. Although the percentage of methane was greater than that in the air of the Red Ash

returns the volume was less, but on the other hand the number of square feet of coal exposed was not quite half as much, or about 700,000, so that the 714 cubic feet of gas represents fully 1 cubic foot to every 1,000 square feet of coal exposed. The daily production was only about one-tenth as much, so the methane was at the rate of 15,600 cubic feet to every ton of coal mined.

BENNETT RETURNS.

The Bennett workings in the Lance mine, shown in figure 23, occupied an area of about 500 acres. Extensive districts are not worked now and the main active workings were a mile south of the main shaft. No attempt was made to sample any considerable part of the return air from these workings but a return was selected that ventilates an area in which considerable mining

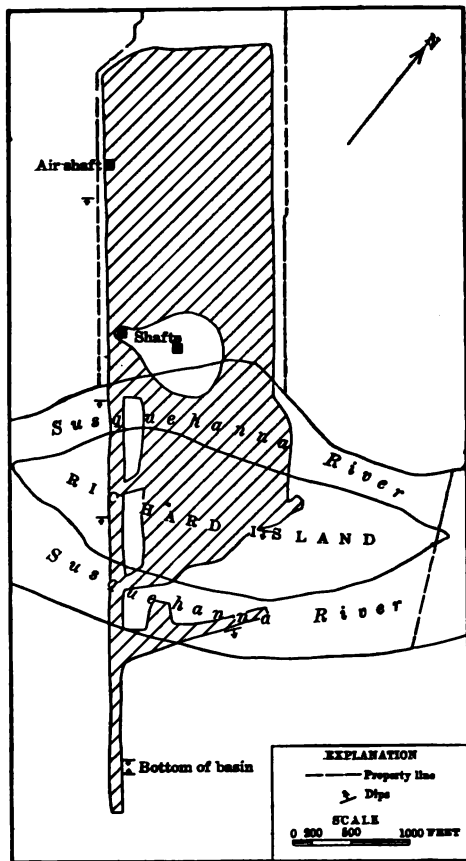


FIGURE 22.—Sketch map of worked area in Ross bed, Lance mine.

was in progress. Two samples also were taken of air receiving the discharge of a large feeder at the head of the incline below a point just north of the corner of Carey Avenue and Division Street, Wilkes-Barre (see fig. 23).

The sample (Lab. No. 1265) of return air from the working area was taken at the point designated A in figure 23. This area is on a low anticline and occupies about 15 acres. The air current had a

volume of 21,663 cubic feet a minute and carried 0.32 per cent of methane, or 70 cubic feet a minute. The average daily production was 55 tons, so that this amount of gas was at a rate of nearly 2,000 cubic feet to the ton of coal mined. The surface of coal exposed in this area was about 90,000 square feet, so that 70 cubic feet of methane indicates the escape of 0.77 cubic feet of methane per minute for every 1,000 square feet of coal exposed. Another sample (Lab. No. 1264) was taken at the point designated *B* in figure 23 near the extreme south end of the workings on No. 4 slope on the south pitch,

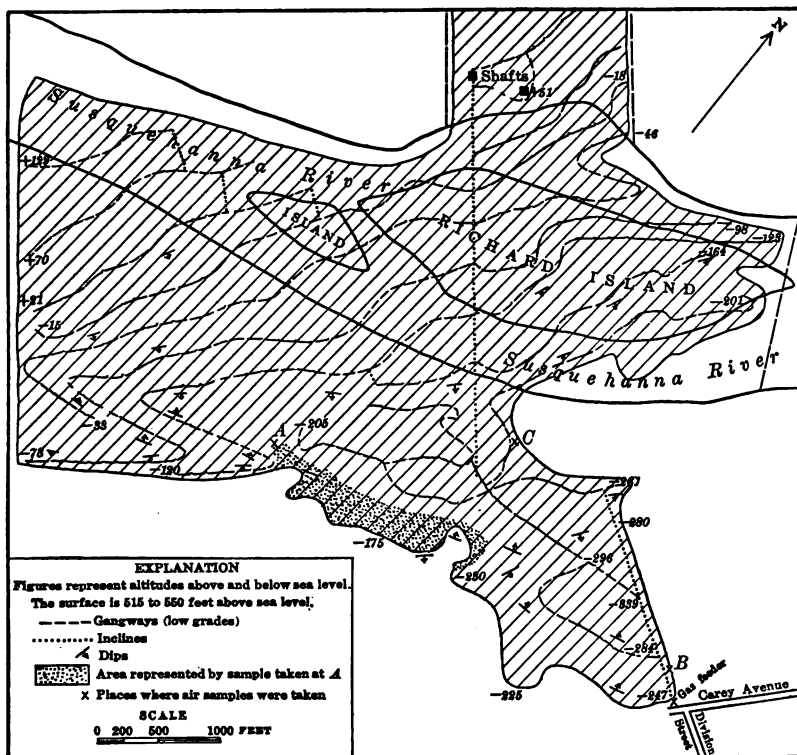


FIGURE 23.—Sketch map of worked area in Bennett bed, Lance mine.

300 feet below a large feeder that had been active for a long time. Between this feeder and the point at which the sample was taken, however, the gas had been diluted by nearly fresh air with a volume of 15,000 feet a minute. The sample contained 2.39 per cent of methane, equivalent to 360 cubic feet a minute, practically all of which is discharged by the gas feeder. The foreman stated that at times the proportion of gas appeared to be much greater. Another sample (Lab. No. 1275) taken at the point designated *C* in figure 23 represents air from the same current sampled at *B*, but here the volume of the return was 41,505 cubic feet a minute as it included air

from other returns and from a split that ventilated six nearby working faces. The methane content of this sample was 1.97 per cent, which indicated a volume of 818 cubic feet a minute, or very much more than that indicated by the sample taken at *B*.

COOPER RETURNS.

The Cooper workings at the Lance mine extend in a broad area south from the shafts under the river and flats, and one gangway

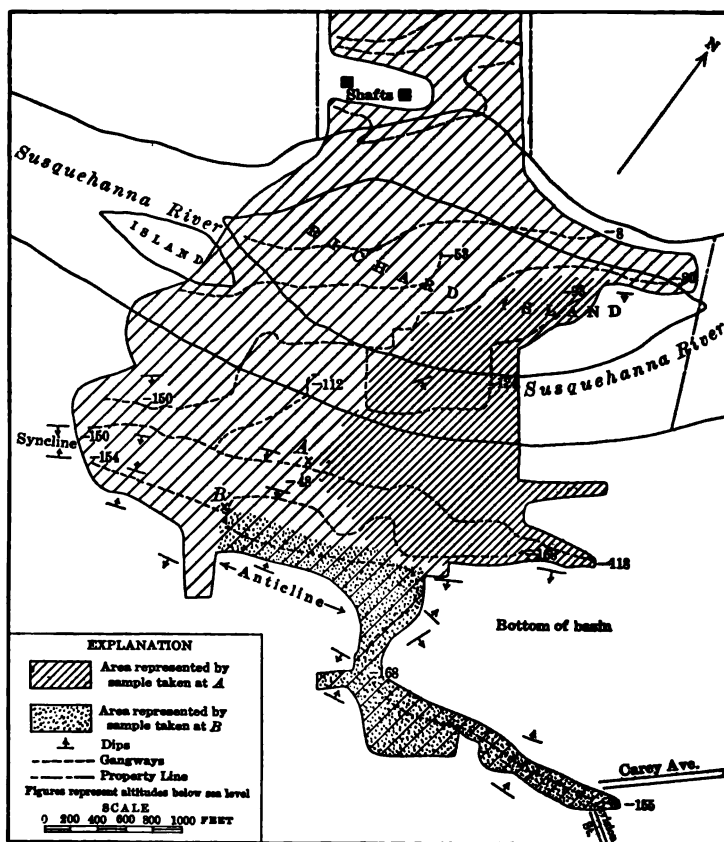


FIGURE 24.—Sketch map of worked area in Cooper bed, Lance mine.

reaches beyond the corner of Carey Avenue and Division Street, in Wilkes-Barre. The Cooper bed is only 50 to 60 feet above the Bennett bed, and the course of the Bennett gangways, shown in figure 24, gives an idea of the structure. There is a long descent into the central basin, and then a rise on the south side. In the central and southern part of the workings the depth below the surface is about 700 feet.

A sample (Lab. No. 1272) of return air from the slopes and an irregular flat basin to the east was taken at the point designated *A*

(fig. 24) in No. 23 tunnel, where the volume of air was 41,500 feet a minute. This air contained 1.04 per cent, equivalent to 431 cubic feet a minute, of methane. About 428,500 square feet of coal was exposed in the area ventilated by this air, so that slightly more than 1 cubic foot of gas was given off for every 1,000 square feet of coal exposed. The daily coal production in these workings averaged 193 tons, and the methane calculated to this factor gives a rate of 3,230 cubic feet to the ton.

Another sample (Lab. No. 1277) was taken at the point designated *B* (fig. 24) from the west return in the No. 11 tunnel, an air course that ventilates all the workings to the extreme south. The region presents an anticline and a syncline in which the beds are considerably disturbed. The volume of the return air was 18,957 cubic

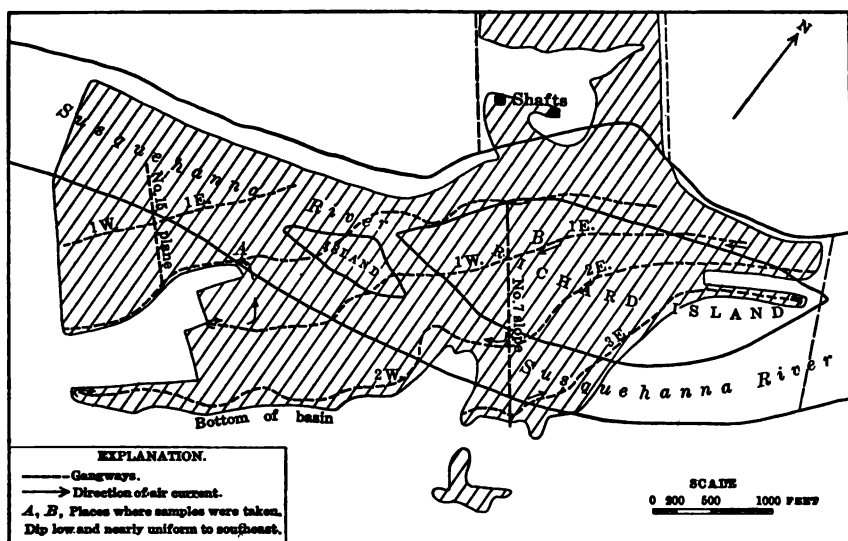


FIGURE 25.—Sketch map of worked area in Five-foot bed, Lance mine.

feet a minute. It carried 2.68 per cent of methane, equivalent to 508 cubic feet a minute, a remarkably large volume of gas when the small extent of the workings is considered. A precise measurement of the area of these workings was not made, but the gas emanation was equivalent to at least 5 cubic feet a minute for every 1,000 square feet of coal exposed. The average daily production from the area was 228 tons, and the volume of methane considered in this relation was at the rate of about 3,200 cubic feet to the ton of coal mined.

FIVE-FOOT RETURNS.

The Five-foot bed has been extensively developed in the Lance mine, the workings extending far under the river and beyond its south bank in places. The principal features are shown in figure 25.

The length of the workings up and down the river was 6,400 feet, and their width south of the shafts was 3,200 feet, comprising an area of about 256 acres. The dip is nearly uniform to the south-east, without noteworthy local disturbances. The bed is 5 feet thick and contains about 6 inches of shale or bone. The system of ventilation is simple. The main return air passes up No. 7 slope almost on the center line of the workings and is derived in nearly equal volumes from gangways from the west, sampled at the point designated *A*, and from the east, sampled at the point designated *B*. The sample taken at *B* did not include air in part of a gangway on the east side, from which a few old chambers had been turned off. The extent of the workings on the two sides was about equal at the time of the author's visit, there being 14 working faces on the west side and 11 on the east side, all of which extended under the river at a uniform depth below the surface. The results of the tests and their relations to various factors are given in the following table:

Results of comparative sampling of air in east and west returns in Five-foot workings, Lance mine.

Source of sample.	Side.	Working places.	Tons per day.	Coal exposed.	Air volume.	Methane.			
						Per cent.	Volume.	To 1,000 square feet of coal exposed.	Per ton.
Point B a.....	East.....	11	89	Sq. feet. 300,000	Cu. feet. 28,900	1.46	Cu. feet. 400	Cu. feet. 1.67	Cu. feet. 6,400
Point A a.....	West.....	14	167	240,000	28,000	.23	70	.23	80

a See fig. 25.

The most notable feature in this table is the great preponderance of methane from the east side, whether calculated to volume, tonnage, or square feet of coal exposed, and the amount of coal mined is only about half as much as on the west side. This excess of gas was not derived from feeders, but appeared to come from coal of the same character and under similar conditions as that on the west side workings.

HILLMAN RETURNS.

The workings in the Hillman bed in the Lance mine, shown in figure 18, are mostly under the river and the flat just south of the river. The measure dips south into the main syncline, but the Hillman workings are all on the north dip. The thickness of the coal bed is 7½ to 8 feet. The return ventilation is from east and west workings through "east" and "west" gangways, emptying into a

central outlet. The area of the workings on the east side is about 25 per cent greater than those on the west side.

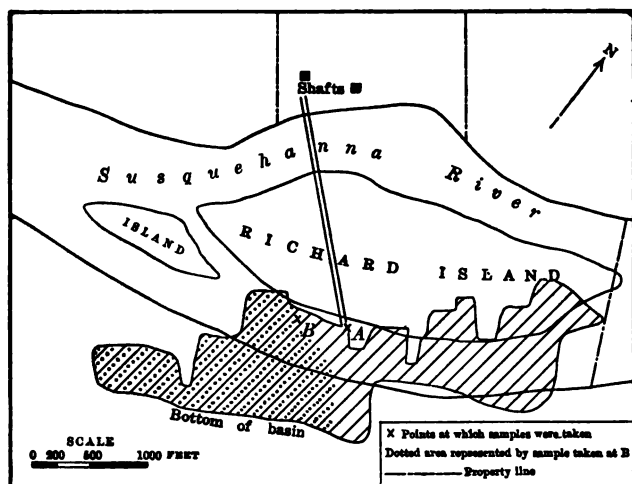


FIGURE 26.—Sketch map of worked area in Hillman bed, Lance mine.

The air was sampled at points designated A and B in figure 26, in the last returns entering on either side of No. 14 tunnel, with results as follows:

Results of comparative sampling of air in east and west returns in Hillman workings, Lance mine.

Source of sample.	Side.	Working places.	Tons per day.	Coal exposed.	Air volume.	Methane.			
						Per cent.	Volume.	To 1,000 square feet of coal exposed.	Per ton.
Point A a.....	East.....	21	208	Sq. feet. 242,000	Cu. feet. 36,800	1.50	Cu. feet. 552	Cu. feet. 2.3	Cu. feet. 3,850
Point B a.....	West.....	14	116	192,000	30,366	.55	167	.9	2,070

a See fig. 26.

The structural conditions on the east and west workings are similar to each other and to those in the Five-foot workings 160 feet below. On the east side the volume of methane is three times that on the west side, whereas the number of working faces is only 50 per cent more and the average daily production is somewhat less than twice as much.

RÉSUMÉ OF CONDITIONS IN LANCE MINE.

The results given on the preceding pages show that in the Lance mine gas occurs under greatly differing conditions. The outflow of

methane from the different districts in the same bed under the same structural conditions varies widely. The greatest amount of gas is in the region south of the shaft under the river and the river flats where the coal beds lie farthest below the surface.

METHANE OUTFLOW FROM HILLMAN AND FIVE-FOOT BEDS.

The Hillman bed is reputed to yield the most gas and this reputation appears just. The return from its east workings, which are of relatively small extent, was found to carry 555 cubic feet of methane a minute, whereas the west return, ventilating a closely similar district, carried only 167. This same difference between the east and west workings is shown in the Five-foot bed not far below, where the east return carried 400 cubic feet of methane a minute and the west return only 70. This difference is seemingly not due to the structure, for the dips are about the same on both sides, nor to the porosity of the overlying beds, which could not account for such a great difference from east to west and between the two beds. Unfortunately it was not practicable to continue this comparison to lower beds, for the fresh workings in the lower beds were not apportioned in the same manner.

OUTFLOW FROM COOPER AND BENNETT BEDS.

The Cooper workings represented by the two samples taken (Lab. Nos. 1272, 1277) lie mostly south and east of the east workings of the Five-foot and Hillman beds. It is stated, however, that the Cooper workings to the west and north were much less gaseous than those to the east and south. The sample taken at *B* (fig. 24) of air from the southernmost workings carried a very high proportion of methane (2.68 per cent), equivalent to 508 cubic feet a minute, whereas the return air from the much more extensive workings in the broad undulating district to the north carried only 1.04 per cent, or 431 cubic feet a minute, which, considering the extent of coal exposed, is not more than one-quarter as much to the square foot of coal. It is stated that the proximity of a fault or break in the beds far to the south is the source of the increased yield of gas and also of the large feeder in the Bennett bed, about 80 feet below. This feeder is of great volume, as shown by the sample (Lab. No. 1264) taken at *B* (fig. 24) in No. 4 Bennett slope, in an air current of 15,000 feet a minute, which contained 2.39 per cent methane, or 360 cubic feet a minute, which was nearly all from the feeder. Air (sample 1265) from a typical working area of Bennett coal taken at *A* (fig. 23), near the bottom of the basin, carried only 0.32 per cent methane, or 70 cubic feet a minute, a volume nearly equivalent to that from the west workings of the Hillman bed that are about twice as extensive. A sample (Lab. No. 1275) of return air taken at *C* (fig. 23) in the bottom of the basin

showed a very large volume of methane, which came in part from the large feeder mentioned above and in part from a gaseous district where a small amount of coal was being mined.

The total volume of methane delivered by the Five-foot upcast, however, was not far from 500 cubic feet a minute, and the amount from the Hillman bed is very close to the sum of the volumes passing at *A* and *B* (fig. 26), which is 722 cubic feet a minute. The content of methane in the air from the main upcast at the mouth of shaft No. 1 was 0.96 per cent, but the volume of this air was not measured. Assuming that it was 200,000 feet, the volume of methane was 1,920 feet a minute, but after the volumes of methane from the Five-foot and Hillman beds are deducted from this figure there remain only 713 cubic feet a minute for the total methane of the Bennett and Cooper returns. This is far too little, as in the No. 2 Bennett slope the volume was 818 cubic feet. However, if the volume of the upcast current be taken as 250,000 feet, the volume of methane becomes 2,400 feet, leaving 1,200 feet for the Bennett and Cooper returns.

OUTFLOW FROM ROSS AND RED ASH BEDS.

The amount of methane coming from the Ross and Red Ash workings was nearly the same, the Ross return showing the larger percentage, but the Red Ash return carrying the larger volume of the gas on account of the greater volume of air, 1,067 cubic feet a minute. Offsetting this in some measure is the fact that the area of coal surface exposed in the thick Red Ash bed was much greater than in the Ross bed, so that the emanation to the square foot of coal exposed was really less in the Red Ash than in the Ross bed. On the other hand, the daily production from the Red Ash bed was nearly ten times as much as was being mined from the Ross workings, so that in the latter there was more than six times as much gas to the ton mined. The samples were taken so near the upcast in both beds that they represented the total gas from each. The extent of the workings was nearly the same in both beds and the structural and other conditions are similar.

GENERALIZATIONS AS TO METHANE EMANATIONS.

In general the amount of gas emanating from the workings of the Lance mine varies greatly in different parts of the various beds, and the local variations in the same bed are not related to structure or to cover. Considerable gas comes directly from feeders or blowers of greater or less size and duration, some of which produce more gas than the regular emanation from a large coal area. Gas also enters some of the workings from the sandstone floor or roof. One notable zone of feeders was developed in a tunnel in which the gas came from a lower split of the Stanton bed through an 8-foot bed of sand-

stone separating the two splits. The large feeder in the south workings in the Bennett bed was emitting 360 cubic feet of methane a minute at the time of the author's visit. If the relative extent of workings be considered, the Hillman bed is the most gaseous except a small area of the Cooper workings. The following table gives some estimates of the number of cubic feet of methane per minute per 1,000 square feet of coal exposed in different parts of different beds.

Volume of methane per 1,000 square feet of coal exposed in different parts of different beds, Lance mine.

Bed.	Part of return.	Volume of methane per minute.
		<i>Cubic feet.</i>
Red Ash.....	Total return.....	0.73
Ross.....	do.....	1.00
Bennett.....	South basin.....	.77
Cooper.....	Central east.....	1.00
Do.....	Southernmost.....	5.00±
Five-foot.....	East side.....	1.67
Do.....	West side.....	.23
Hillman.....	East side.....	1.00
Do.....	West side.....	2.30

In reading these figures one should consider that the amount of gas from working faces is much greater than from old pillars and gangways, the relative extent of which was not ascertained. The relation of methane emanation to amount of coal mined gives some interesting figures, but their significance is greatly diminished by the fact that after the mine had been shut down nearly a month in April, 1912, the volume of gas had not greatly diminished in any of the workings, and in some it was the same or greater. The figures in the following table show the relation of volume of methane to average daily output in January, 1911:

Volume of methane per ton of coal mined at Lance mine.

Bed.	Part of return.	Average daily production.	Volume of methane.	
			Per minute.	Per ton mined, approximate.
		<i>Tons.</i>	<i>Cubic feet.</i>	<i>Cubic feet.</i>
Red Ash.....	Total return.....	643	1,067	2,400
Ross.....	do.....	66	714	15,600
Bennett.....	South basin (No. 25 tunnel).....	55	70	2,000
Cooper.....	Central east (No. 23 tunnel).....	193	431	3,230
Do.....	Southernmost (No. 11 tunnel).....	228	508	3,200
Five-foot.....	East side.....	89	400	6,400
Do.....	West side.....	167	70	60
Hillman.....	East side.....	208	552	3,850
Do.....	West side.....	116	167	2,070
All beds.....	Two shafts.....	1,930	3,701	2,761
Red Ash and Ross.....	North shaft.....	709	1,781	3,617
All except Red Ash and Ross beds.....	South shaft.....	1,221	1,920	2,564

INFLUENCE OF RIVER DEPOSITS.

As most of the Lance workings south of the shaft lie below river deposits of varying thickness, it appeared possible that these might affect the occurrence of gas in the beds below. Part of the deposits are thick and contain considerable clay which would tend to retard the general outflow of gas from the coal. Special attention was given to this feature, and as there were many bore holes the extent

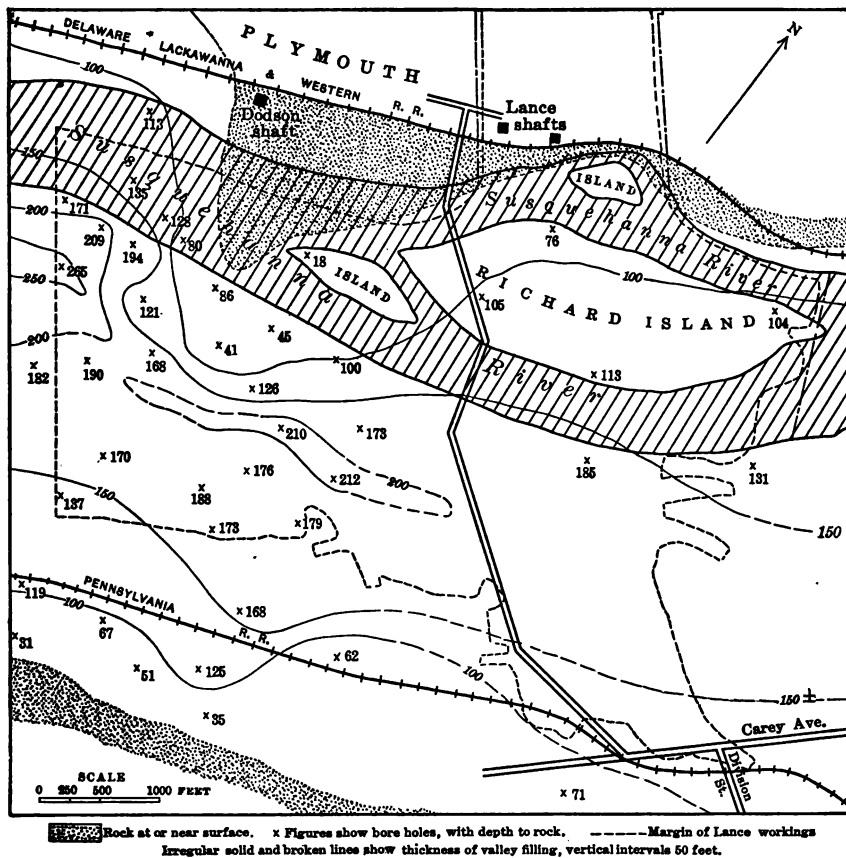


FIGURE 27.—Sketch map showing thickness of valley filling, Lance mine.

and thickness of the valley filling could be ascertained. The principal data bearing on this are given in figure 27.

This map shows a deep central channel with the rock rising to or near to the surface on either side. At the deepest place the filling is more than 250 feet thick, and there is a wide area in which it is more than 150 feet thick. Although sand and gravel are the principal materials, there is enough clay to affect permeability decidedly. A promontory of rock southwest of the Lance shafts is a notable feature and the sandstone which comes to the surface must be the avenue of

escape for considerable gas. It is a fact that the most gaseous parts of the Lance workings are under the center of the channel southeast of Richard Island; but, on the other hand, the less gaseous parts of the present Five-foot, Hillman, Cooper, and Bennett workings are under an equally thick valley filling. It is stated that the workings of the higher beds in the vicinity of the rock promontory above mentioned were not especially gaseous, but not materially less so than in some districts more deeply covered by the valley filling. There are in the coal measures thick beds of fire clay that appear to be just as impervious as the clay in the valley fillings, and no doubt these are a most important factor in impeding the escape of gas to the surface.

SOUTH WILKES-BARRE MINE.

In Mine No. 5 of the Lehigh & Wilkes-Barre Coal Co. a series of return-air samples was collected to ascertain the relation of certain structural features to the occurrence of methane. The coal measures are here flexed into anticlines and synclines of moderate steepness, most of the dips being from 5° to 45° . There are, however, several local rolls or so-called "faults" in which the dips are steeper. The principal beds worked are the Baltimore, Hillman, Abbott, and Kidney. Owing to the folding, the depths of the beds vary greatly, some of the Hillman workings in the northern part of the mine being more than 1,100 feet below the surface. There is considerable gas throughout the mine, but its volume and manifestations have varied greatly from time to time. The Hillman bed was producing the most gas at the time of sampling, but it was stated that the Baltimore bed, which was not being worked so extensively as formerly, had many very gaseous places and occasional transient blowers.

RETURN-AIR SAMPLES.

In taking samples of return air in this mine the Abbott and Hillman workings were selected because these beds were especially gaseous and the system of ventilation was such that representative samples could be obtained readily. Moreover, the workings extended through areas presenting considerable variation in structure. The samples were analyzed, with the following results.

Results of analyses of samples of return air from Abbott, Hillman, and Stanton workings, South Wilkes-Barre mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Location on map.	Volume of air per minute.	Velocity of air per minute.	Proportion of methane in mine air.	Volume of methane per minute.
1209	Abbott.....	No. 7 slope, 300 feet south of No. 5 shaft.	A	<i>Cubic feet.</i> 20,180	<i>Feet.</i> 240	<i>Per cent.</i> 1.14	<i>Cubic feet.</i> 230
1208	do.....	No. 4 tunnel outlet, 80 feet north of No. 5 shaft; station 1244.	B	44,676	438	.44	197
1218	do.....	No. 4 tunnel outlet, 80 feet north of No. 5 shaft; station 1244, 2d outlet.	C	72,420	710	.42	304
1214	Abbott, east.....	Return from No. 17 tunnel.	D	a 21,175	605	.32	68
1216	do.....	Return from No. 11 tunnel.	E	b 17,550	585	.18	32
1215	Hillman and Stanton, west.	Outlet of No. 4 tunnel near station 105.	F	30,240	210	1.43	432
1213	do.....	No. 4 tunnel, east of station 866.	G	c 18,450	246	.67	124
1217	Stanton and Hillman..	No. 4 tunnel; station 853.	H	33,264	336	.95	316
1266	Total upcast of all beds.	Fan No. 1.....		315,000	1,226	.72	2,269

a Nearly fresh air from No. 3 shaft and Hillman and Kidney gangways; entered Abbott workings through No. 17 tunnel, at the end of which it was split, and ventilated 7 working faces.

b Air entered at No. 3 shaft, passed along Kidney and Hillman gangways, then up plane and along No. 11 tunnel into Abbott workings.

c Air passed from No. 3 shaft down No. 4 tunnel, and through 9 working faces at 70 feet below sea level.

The total volume of upcast air of the South Wilkes-Barre mine was sampled at fan No. 1, on shaft No. 5, where it was approximately 315,082 cubic feet a minute. The methane content was 0.72 per cent, or 2,269 cubic feet a minute. This sample represents air from all the workings in the Baltimore and in the higher beds.

AIR SAMPLES FROM THE ABBOTT RETURNS.

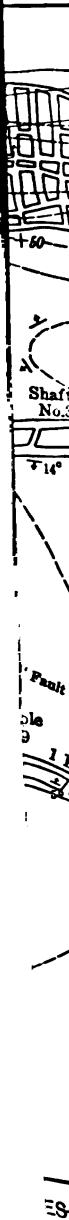
The Abbott workings at the South Wilkes-Barre mine are mostly east, north, and south of the shafts. The north workings are in a long, deep basin, in part of which the measures extend somewhat below sea level, or 500 to 600 feet below the surface. The length of the workings in this basin was about 3,600 feet along an east and west line, the easternmost chambers being at Hazel Avenue, a short distance south of Main Street, Wilkes-Barre. The dips in this basin are mostly 6° to 40°, many of them being more than 15°. The surface is covered by a thin body of sand and gravel deposited by the river.

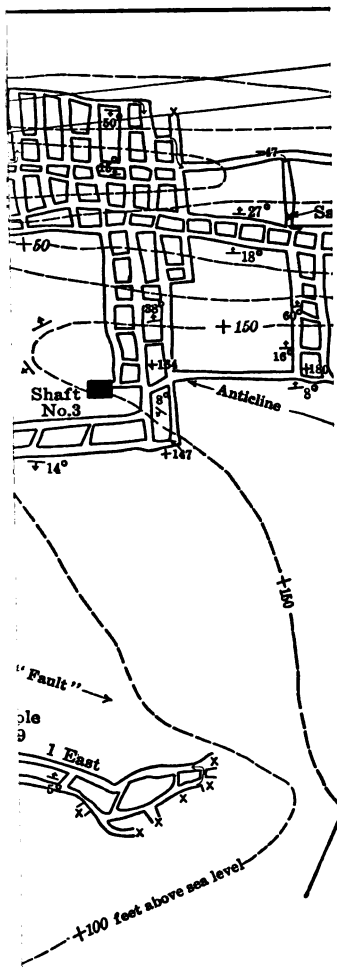
The south workings are in strata which rise gently to the southward, and are about 500 feet below the surface, or 30 to 145 feet above sea level. The surface is hilly and mostly rocky, with a scattered covering of glacial till. The south workings had been extended to a point about 3,600 feet southwest of shaft No. 5. In the Abbott

workings the strata are not broken by true faults, but there are many local rolls in the beds, and in places considerable crushing and shattering, especially in the steeper slopes into the north basin. In general the workings are gaseous, especially in the breasts where coal is freshly exposed. The average thickness of the coal bed is about 6 feet.

The return air from the Abbott workings was sampled at several places so as to compare the amount of methane given off in various districts where depth or structural conditions differ. The air enters at shaft No. 3 and shaft No. 5; the intake from No. 3 entered by airways from other beds and was distributed through tunnels. The principal features are shown in Plate III. Sample 1209 was taken from the return from the extensive southwest workings in the Abbott coal, in all of which the dip of the beds is north at angles, mostly between 10° and 30° , with local variations. The air enters by the No. 5 shaft and at the place where sample 1209 was taken, 300 feet from the shaft upcast, it had a volume of 20,160 cubic feet a minute. As shown in Plate III, 31 faces were being worked, all of them up the dip. The return carried 1.14 per cent methane, or 230 cubic feet a minute. The length of the workings being about 58,600 feet and the height about 6 feet, there was about 351,600 square feet of coal exposed. Therefore 0.65 cubic foot of methane a minute was given off for every 1,000 square feet of coal surface. The average daily output from these workings was 410 tons, so that the methane was at the rate of 808 cubic feet to the ton of coal, a low rate as compared with the amount in most of the Lance workings.

Samples 1208 and 1218 were taken in two airways just north of shaft No. 5, or the main upcast; the samples represented the return air from all the north workings in the Abbott bed. These workings are in the deep, closely compressed basin north of the shafts and extending east as far as Hazel Avenue, Wilkes-Barre. The beds dip into the basin at angles of 8° to 60° and in places are considerably crushed. There were only a few breasts being worked at the time, and these were at the points shown in Plate III. The air enters the mine at No. 3 shaft; part of it goes directly to the Abbott workings and part indirectly through gangways and tunnels in other beds. The volume of the return air of the two airways at the time of sampling was 117,096 cubic feet a minute. The proportion of methane was 0.42 per cent in one airway and 0.44 per cent in the other, indicating a total volume of 501 cubic feet of methane a minute. The length of coal exposed in passageways and chambers in the Abbott workings ventilated by this air was 46,000 feet and as the average height is 6 feet the total area of coal exposed was 276,000 square feet. A calculation from these figures gives 1.8 cubic feet of methane a minute for each 1,000 square feet of coal exposed.





ES-BARRE MINE, PA.

The air represented by sample 1214 entered the mine at the No. 3 shaft, passed along some Hillman and Kidney gangways and then through No. 17 tunnel to the point designated "A" in Plate III, where it was split to ventilate seven working faces in the deep basin lying north of the shaft. These workings extend east for about 1,300 feet to the point where the sample was taken, and were only moderately gaseous. The air current measured 21,175 cubic feet a minute and contained 0.32 per cent methane, or 68 cubic feet a minute. As about 66,000 square feet was exposed in the workings ventilated, slightly more than 1 cubic foot of methane to 1,000 square feet of coal is indicated.

A sample of this air taken by H. I. Smith in February, 1912, in No. 17 tunnel east of station 866 contained 0.89 per cent of methane. The air volume was 6,200 feet and its methane content was 51 cubic feet a minute.

Sample 1216 was taken in the east Abbott return, 500 feet east of shaft No. 3. This return carried the air from the basin extending for nearly a half mile east to Hazel Avenue, in which no active mining had been done for a year and a half. The air current had a volume of 17,550 cubic feet a minute and contained only 0.18 per cent of methane, equivalent to 32 cubic feet a minute. About 141,000 square feet of coal was exposed, so that the outflow of methane per 1,000 square feet of coal exposed was only 0.227 cubic foot per minute.

AIR SAMPLES FROM THE HILLMAN RETURNS.

The Hillman workings in the northern part of the South Wilkes-Barre mine yield so much gas that at times mining operations have had to be discontinued. The workings are mainly in two deep basins, the northern one extending to 625 feet below the surface (90 feet below sea level) and the southern one to 850 feet below the surface (more than 300 feet below sea level). An anticline 500 feet or more high separates the basins, and there is another anticline at shaft No. 3 on the south side of the south basin. The relations and the extent of the workings (in November, 1911) are shown in Plate IV, which also bears contour lines and dip marks showing the structure. The area was ventilated by air from the downcast at shaft No. 3, which passed along an airway a short distance and then through the long No. 4 tunnel. Samples 1215 and 1216 represent the total return air from the northern basin, whereas sample 1217 mainly represents air from the workings on part of the north slope of the south basin. Sample 1215 was taken in one of the main returns; this air contained 1.43 per cent methane, equivalent to 432 cubic feet a minute. Sample 1213 was taken in an airway in the north basin from another return, which ventilated nine working

faces. The return air contained 0.67 per cent of methane, 124 cubic feet a minute, which, added to the amount at the place where sample 1215 was taken, makes 556 cubic feet for the total volume from the northernmost workings.

If the length of coal face in these workings, about 14,500 feet, is multiplied by the thickness of the bed, 7 to 8 feet, the number of square feet of coal exposed becomes 108,700. Five hundred and fifty-six cubic feet of gas a minute from this area is at the rate of slightly more than 5 cubic feet for each 1,000 square feet of coal surface. Of course the rate of emanation varies greatly and is much more in the fresh working faces than in pillars and old chambers. The average daily coal production from this district was 94 tons, so that the methane emanation calculated to this tonnage is 8,534 cubic feet to the ton.

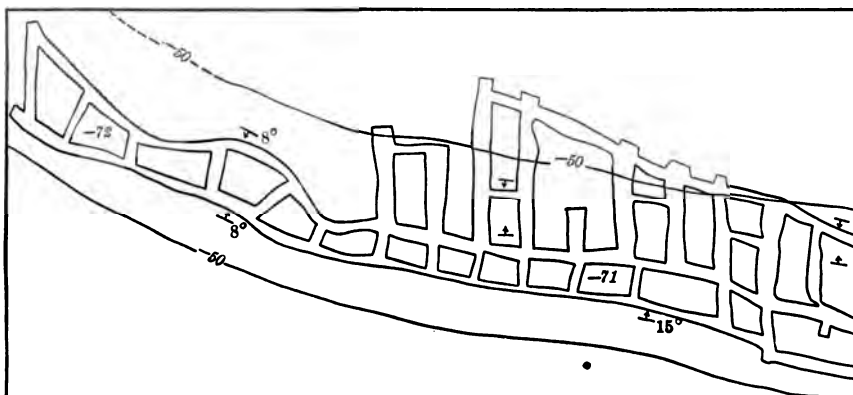
Sample 1217 represented a decidedly gaseous area in the Hillman workings lying a short distance north of shaft No. 3 and about 500 to 650 feet underground. The strata dip 15° to 30° south into a deep basin. The air passed from No. 3 shaft along the Hillman airway through No. 3 and No. 4 tunnels and about 800 to 1,000 feet of gangways, being diluted somewhat by air from a small area of workings in the Stanton bed where two places were being worked. No work was being done in this Hillman area. The length of faces in it was about 9,000 feet and their average height 7 to 8 feet, so that the coal surface exposed was about 67,500 square feet. The outflow of gas from this area being 316 cubic feet a minute and 250 feet of this being assumed to come from the Hillman workings, the volume given off for every 1,000 square feet of coal exposed was 3.7 cubic feet a minute, or decidedly less than in the basin represented by samples 1215 and 1213. A sample of this air, taken early in April, 1912, carried 2.25 per cent of methane, and another sample, taken in No. 4 tunnel, 2.36 per cent.

The mine foreman stated that the Hillman bed appeared to be as gaseous where undisturbed as in or near slips or breaks or along anticlinal and synclinal axes. The very gaseous places in the north workings are along the two sides of a low anticline. A sample of gas from a cavity in the roof over a fall in the No. 5 slope was found to carry 93.88 per cent methane, 0.25 per cent carbon dioxide, 2.17 per cent oxygen, and 3.70 per cent nitrogen.

RÉSUMÉ OF CONDITIONS IN SOUTH WILKES-BARRE MINE.

In this mine the most gaseous district, the north Hillman workings, was producing 556 cubic feet of methane a minute, or more than 5 cubic feet a minute for every 1,000 square feet of coal exposed. The return from the Hillman workings in the basin near the shaft (Lab. No. 1217) carried nearly 1 per cent methane, representing a

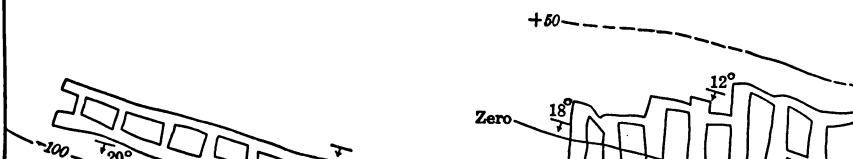
BUREAU OF MINES

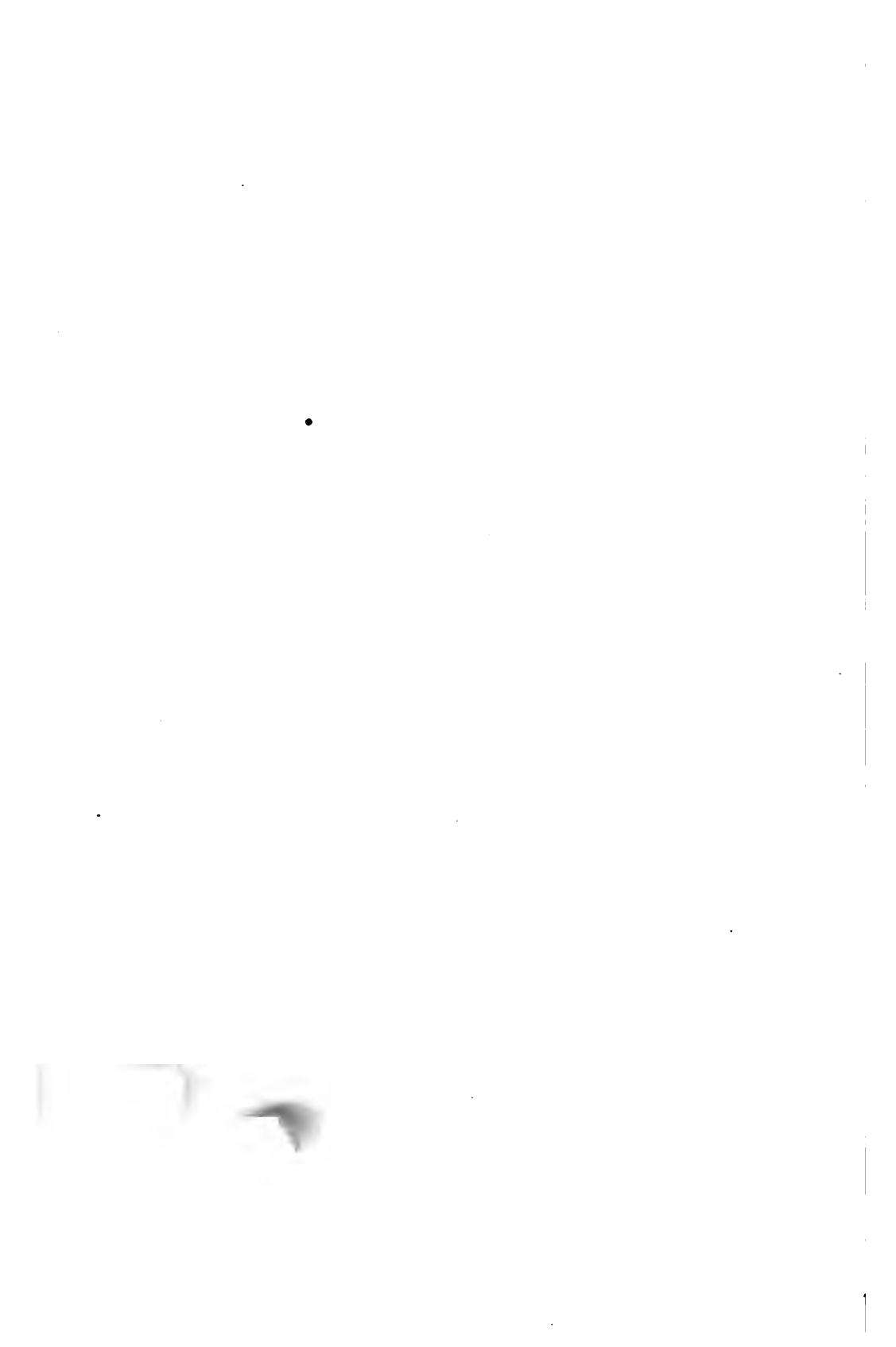


↘ Dips

The figures and contour lines show altitudes
above and below sea level

SCALE
0 100 200 300 400 500 FEET





volume of 3.7 cubic feet a minute for every 1,000 square feet of coal. The difference in the amount of methane produced in these two areas might be entirely due to the active mining in the north basin, but according to the testimony of the mine foreman this was not the case. Neither could the variation have been due to differences in the geologic structure or to the thickness and tightness of cover, for these were too nearly alike in both areas.

There is much less gas in the Abbott bed 220 feet above the Hillman, although both coals are regarded as of the same general class and have nearly the same thickness. In the basin over the one represented by sample No. 1217 the methane emanation (see Lab. No. 1214) was only 0.32 per cent, or 1 cubic foot a minute to every 1,000 square feet of exposed coal, less than one-third as much as in the Hillman workings. Similarly, the return air (sample 1216) from the Abbott workings, east of the shaft, and on the eastern extension of the same basin from which sample 1214 was taken, carried only 0.18 per cent methane, or only 0.23 cubic feet a minute for every 1,000 square feet of coal surface. The difference in the amount of gas at the two places is believed to be due to the fact that the air from which sample 1214 was taken ventilated active working faces (seven), whereas the area represented by sample 1216 had not been worked for some time.

The Hillman workings southwest of the shaft were not especially gaseous as a whole, and the air, which was not sampled, probably carried much less gas than that from the north workings represented by samples 1215, 1213, and 1217. The Abbott workings in the area southwest of the shaft gave off twice as much gas as those to the north. This is shown by the methane content of sample 1209, from a return of nearly the same volume as the one represented by sample 1214, but the former carried 1.14 per cent methane, or 0.65 cubic feet a minute for every 1,000 square feet of coal surface. However, in this district there were 31 working faces, as against 7 in the district from which sample 1214 was collected, which may account for part of the difference.

DORRANCE MINE.

The Dorrance mine is one of the newer collieries of the Lehigh Valley Coal Co. It is situated in the northern edge of Wilkes-Barre, and its workings extend north under the Susquehanna River and the flats beyond nearly to Kingston Corners. It has been an exceptionally gaseous mine, and parts of it have had to be operated with special care on that account. Considerable attention was given to this mine because of the varied conditions that could be investigated. The workings show diversity of geologic structure, there being a general deep syncline north of the river and several subordinate undulations. The gaseous areas were irregular in extent, and much gas was being or had been derived from "blowers." The Hillman workings have had

the most noticeable gas manifestations, but the analyses of return air demonstrated that the smaller Five-foot workings produced somewhat more gas for each square foot of coal exposed.

SUCCESSION AND STRUCTURE OF BEDS.

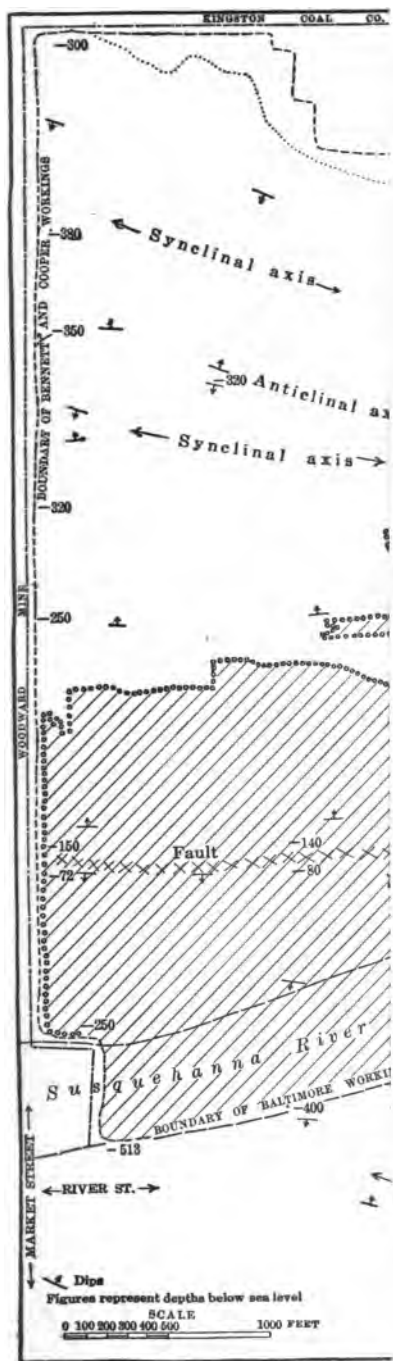
The area being mined at Dorrance included two deep synclines containing about 1,000 feet of strata and all the coal beds represented in the northern basin, their aggregate thickness being more than 75 feet (see fig. 18). At the base is the Red Ash bed, which attains a thickness of 14 feet over a wide area. The two Ross beds, 3 to 4 feet thick, are nearly 100 feet higher, and the Baltimore bed is 250 to 400 feet above the Red Ash bed. The Baltimore bed, which averages 14 feet in thickness, splits into two beds, the Cooper and the Bennett, not far north of the Susquehanna River. These beds finally become separated by a thick wedge of sandstone near Wyoming Avenue, in Dorranceton. The Five-foot bed lies 200 feet above the Baltimore (Cooper), and the Hillman, 6 feet thick, is 350 feet above. In the next 150 feet are the Bowkley and Abbott (Snake Island) beds, each about 5 feet thick, and there are other thinner coal beds still higher up. The surface north of the Susquehanna is a wide river flat, mantled with about 100 feet of sand, gravel, and clay. The proportion of clay is sufficient to constitute a moderately impervious capping competent to impede the escape of gas, though at certain points considerable gas escapes to the surface.

The structure in the Dorrance mine area is shown in Plate V by dip marks, axial lines, and elevations on the Baltimore coal bed. In general, the workings are in the deep central part of the main syncline of the basin. There are several minor folds within this basin which have axes essentially parallel to the axis of the main trough. One of these is an anticline extending across the worked area with its axis under the north bank of the river and developing into a fault to the west. The strata and its southern side descend about 500 feet into a deep syncline that passes south of the Dorrance shaft. The north slope of the anticline descends into a broad basin 100 to 200 feet deep, separated into two synclines by a long, narrow anticline that in part is faulted near its crest. This anticline is the so-called "fault" that crosses the Pettebone workings to the east.

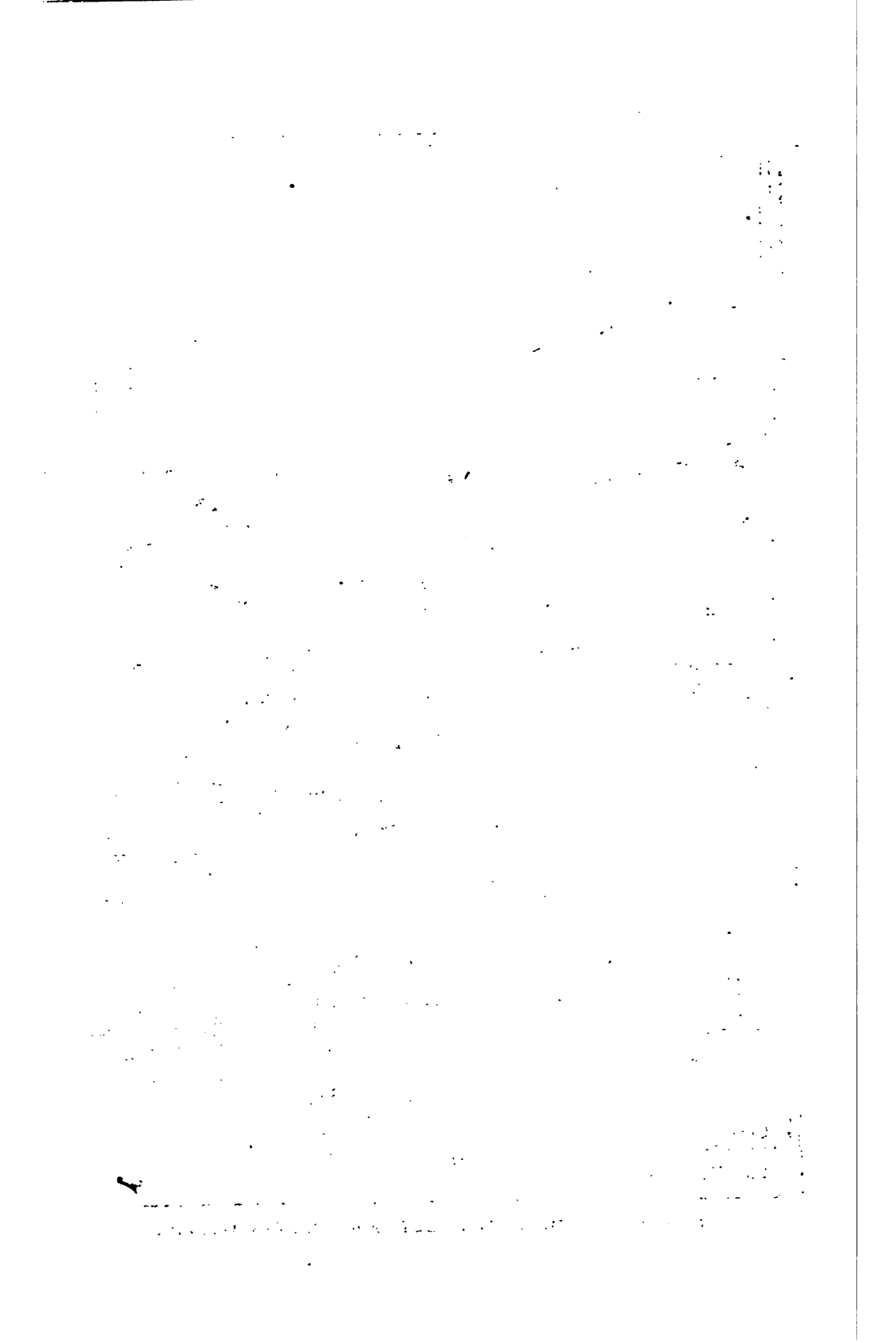
RETURN-AIR SAMPLES.

The main returns of the Dorrance mine were sampled in December, 1910, and some of them were sampled again in the following month. Additional samples were collected by H. I. Smith early in April, 1912. The analyses of these samples are given in the table following. Analyses of samples taken by the author late in April, 1912, after mining had been suspended for nearly a month, are given on page 144.

BUREAU OF MINES



MAP OF BALTIMORE,



Analyses of return air in Dorrance mine.

[G. A. Burrell, analyst.]

Lab. No.	Date.	Return.	Place of sampling.	Air current.		Methane.		CO ₂ .
				Velocity per minute.	Volume per minute.	Proportion in mine air.	Volume per minute.	
1219	Dec. 16, 1910	Red Ash.....	Rock slope to Baltimore.	<i>Cu. ft.</i> 865	<i>Cu. ft.</i> 87,600	<i>Per ct.</i> 0.99	<i>Cu. ft.</i> 868	<i>P. ct.</i>
1298	Jan. 25, 1911	do.....	do.....	709	95,715	1.07	1,024	0.09
2531	Apr. —, 1912	do.....	do.....	647	78,900	1.37	1,080	.10
1300	Jan. 25, 1911	Baltimore-Red Ash - Five-foot.	Foot of Baltimore shaft, north side.	1,428	197,540	.49	968	.08
2534	Apr. —, 1911	do.....	do.....	1,110	139,900	1.02	1,427	.45
1299	Jan. 25, 1911	do.....	Foot of Baltimore shaft, south side.	1,428	101,844	.64	652	.10
2533	Apr. —, 1911	do.....	do.....	1,300	159,000	.73	1,161	.06
1221	Dec. 16, 1910	Five-foot.....	On plane to Baltimore.	190	13,870	1.38	191	
2541	Apr. —, 1912	do.....	do.....	153	18,700	.84	157	.31
1212	Dec. 16, 1910	Hillman.....	At shaft, split "A".....	820	96,600	1.12	1,082	
1211	do.....	do.....	At shaft, split "B".....	756	75,600	.92	695	.10
2538	Apr. —, 1911	do.....	do.....	598	53,800	.55	296	.07
2540	do.....	Hillman - Abbott-Kidney.	At shaft, west side.....	562	66,300	.64	424	.07
1220	Dec. 16, 1910	Hillman.....	At west boundary, split "A".....	241	20,485	.99	203	
1210	do.....	do.....	At west boundary, split "B".....	178	16,109	.88	142	
2537	Apr. —, 1912	Kidney.....	150 feet from Hillman shaft.	432	18,100	1.26	242	.05

The total emanation of gas from the Dorrance mine was not ascertained, but on December 16, 1910, the Red Ash, Five-foot, Baltimore, and Hillman returns, which comprise the greater part of the working area, were giving off about 3,390 cubic feet of methane a minute. Early in April, 1912, the amount was 3,308 feet, and later in April, when mining had been suspended for nearly a month, the same returns carried 3,212 cubic feet of methane a minute.

AIR SAMPLES FROM RED ASH RETURNS.

Samples 1219, 1298, and 2531, which represent the entire return air from the Red Ash workings, were taken in No. 13 rock slope, through which this air was carried up to the Baltimore workings. One sample taken in December, 1910, contained 0.99 per cent methane, so that the return was passing 868 cubic feet a minute. A month later the proportion was 1.07 per cent, equivalent to a volume of 1,024 cubic feet a minute. The workings are under and north of the river, due north of Dorrance breaker, as shown in figure 28. They extend 2,800 feet west from the east line of the property and two of the slopes reach 1,000 and 1,200 feet north of the river. These slopes pass over a low anticline and some distance down its north side.

The depth beneath the surface varies from about 1,000 feet on the crest of the anticline to 1,050 and 1,150 feet in the synclines north and south. The average thickness of the bed is about 14 feet. The

length of faces, pillars, gangways, and other coal surfaces in the workings was about 72,800 feet and as the average height is 14 feet there were 1,019,200 square feet of coal exposed. At the time of sampling there were 50 working faces, some of which were also mined on the night shift, so that there was a large area of fresh coal surface exposed. The gas is believed to be given off uniformly, the greatest amount coming from the working faces. If the total emanation of methane, 1,024 cubic feet a minute, be compared with the area exposed, it is equal to slightly more than 1 cubic foot a minute for each 1,000 square feet of coal surface, but doubtless the outflow was

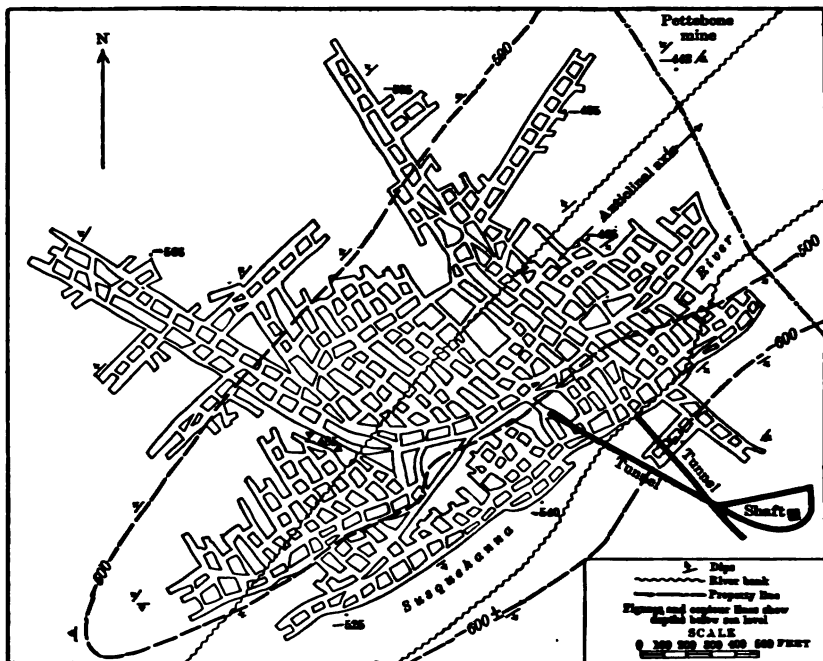


FIGURE 28.—Map of Red Ash workings in Dorrance mine, Wilkes-Barre, Pa.

many times as much from working faces as from pillars. The production at time of sampling was 620 tons a day, including waste, so that the methane emanation was somewhat less than 2 cubic feet per minute for every ton mined.

COMPOSITION OF RED ASH COAL.

A sample of Red Ash coal was taken from a typical working face on January 2, 1911, 600 feet northwest of the main Dorrance shaft. The sample, which represents the entire face, was analyzed in the laboratory of the Bureau of Mines at Pittsburgh, with the following results.

Analysis of Red Ash coal from Dorrance mine, Wilkes-Barre, Pa.

[A. C. Fieldner, analyst.]

	Condition.			Referred to coal "moisture and ash free."
	Air dried.	As re- ceived.	Moisture free.	
Proximate analysis:	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Moisture.....	a 1.01	1.31		
Volatile matter.....	5.70	5.68	5.76	6.21
Fixed carbon.....	86.13	85.87	87.01	93.79
Ash.....	7.16	7.14	7.23	
Ultimate analysis:				
Hydrogen.....	2.33	2.35	2.23	2.40
Carbon.....	87.02	86.76	87.91	94.76
Nitrogen.....	.68	.68	.69	.74
Oxygen.....	2.39	2.65	1.51	1.64
Sulphur.....	.42	.42	.43	.46
Ash.....	7.16	7.14	7.23	
Calorific value determined:				
Calories.....	7,677	7,654	7,756	8,360
British thermal units.....	13,819	13,777	13,961	15,048
Calorific value calculated from ultimate analysis:				
Calories.....		7,715		
British thermal units.....		13,887		

a Loss in air drying, 0.30 per cent.

AIR SAMPLES FROM BALTIMORE RETURNS.

The samples of air (Lab. Nos. 1300, 2534, 1299, and 2533) from the Baltimore workings were taken in the two outlets at the foot of the Baltimore upcast shaft. The air comprises not only the total return air from the Baltimore workings but also that of the Red Ash and Five-foot returns. The total volume of methane carried was 1,620 cubic feet a minute, but if the values for the Red Ash (1,024 cubic feet) and the Five-foot (about 250 cubic feet) be deducted, there remains only 346 cubic feet to the minute as the volume from the Baltimore workings. These workings extend under a large portion of the wide district lying between the Pettebone mine and North Market Street, Wilkes-Barre, and from the south side of the river north nearly to Rutter Street, as well as a small area southeast of the Dorrance shaft. In the north half of the area the Baltimore coal separates into two beds, the upper (7 feet thick) being known as the Cooper bed and the lower (6½ feet thick) as the Bennett, which are worked separately. The area of the workings, which are shown in figure 28, was approximately 1,000 acres, of which probably about one-third was pillars and two-thirds gangways and chambers. The workings were ventilated by about 200,000 cubic feet of air a minute, which, seemingly, carried only about 0.2 per cent methane. There were large areas of old workings, but considerable coal was being mined in places.

The workings extend over the prominent anticline just north of the river, the crest of which to the west develops into a 60 to 80 foot

fault, and to the north descend 200 feet into a broad syncline traversed by a low anticline. The depth below the surface is mostly 600 to 800 feet and much of the area is covered by 50 to 100 feet of sand, gravel, and clay deposited by the Susquehanna River.

AIR SAMPLES FROM FIVE-FOOT RETURNS.

The return from the Five-foot workings could be sampled to advantage only at the head of No. 10 plane, or after it had ventilated about two-thirds of the worked area. As shown in Plate V this area, about 600,000 square feet, extends along the north side of the Susquehanna River just north of Dorrance breaker for about 2,000 feet east and west and 600 feet north and south. The beds dip gently 5° to 10° north into the main basin and are approximately 500 feet below the surface. The return sampled ventilated about 20 working faces which, with gangways and other spaces, had a length of about 15,000 feet. As the coal bed is 5 feet thick, 75,000 square feet of coal was exposed. The return air (sample 1221) carried 191 cubic feet of gas a minute, or about 2.5 cubic feet a minute for every 1,000 feet of coal surface. The amount of coal being mined was nearly 110 tons a day, including waste, so the methane was at the rate of about $1\frac{1}{2}$ cubic feet a minute per ton.

AIR SAMPLES FROM HILLMAN RETURNS.

It was practicable to sample the Hillman returns at only two places, one in the main return at the foot of the upcast shaft, and the other in the main return near the west boundary of the property. The area of the Hillman workings in the Dorrance mine is shown in Plate V. The workings comprise a broad belt under the river and an extensive district to the northwest, covering in all about 220 acres. They extend across the prominent anticline previously mentioned and down into the syncline to the north, and the northernmost gangway passes over the next anticline north. The depth beneath the surface is mostly from 300 to 400 feet and except near the Dorrance shaft the entire area is covered by about 100 feet of sand, gravel, and clay deposited by the river. The return air from the Hillman workings comes to the foot of the upcast shaft through two airways. One, carrying 1,082 cubic feet of methane (see sample 1212) a minute, is from the eastern and southern parts as far as the downcast at Thompson Street, Wilkes-Barre. To the east some miners were robbing pillars in an area filled by flushing. The thickness of the bed is about $6\frac{1}{2}$ feet.

One split ventilated a small area of workings in the Bowkley coal bed and then passed out the upcast, but this split carried little of the Hillman gas. The second airway, sampled at the foot of the main

upcast (sample 1211) and carrying 695 cubic feet of methane a minute, received two splits: one drained new workings north of the river, where about 50 miners were employed, and a large area of old workings; the other was from workings that extended under the river and to the north as far as the west boundary.

The length of coal surface exposed in all chambers and gangways, as measured on the mine map, was approximately 160,333 feet, and as the average thickness of the bed is slightly over 6 feet, about 1,000,000 square feet of coal was exposed without deducting for certain areas that had been filled by flushing. As the total volume of gas in the two airways at the main upcast was 1,777 cubic feet a minute, 1.78 cubic feet of methane a minute was given off for every 1,000 square feet of coal exposed. Much more of this gas undoubtedly came from new workings than from old pillars and abandoned chambers.

The return air from the Hillman workings along the west boundary of the area was in two airways and these are represented by samples 1220 and 1210. One carried 203 and the other 142 cubic feet of methane a minute, a total of 345 cubic feet. This air finally joined other splits and was included in the air from which sample 1211 was taken. The extent of the area represented by this return air was large but was not ascertained.

RÉSUMÉ OF CONDITIONS IN DORRANCE MINE.

The amount of gas emanating from the different coal beds in the Dorrance mine varies widely and as the structural conditions are closely similar throughout, these variations are not related to the structure.

The Hillman workings, which present structural conditions similar to those of the Baltimore workings, but occupy a much smaller area (220 acres), were yielding 1,777 cubic feet of methane, equivalent to 1.78 cubic feet a minute to every 1,000 square feet of coal exposed. The Baltimore and Cooper-Bennett workings, having an area of nearly 1,000 acres and coal twice as thick, have a rate only about one-fiftieth as great, although certain places in these extensive workings are very gaseous. The Red Ash workings with only 84 acres mined but very thick coal gave slightly more than 1 cubic foot of methane a minute for every 1,000 square feet of coal exposed, or nearly 2 cubic feet a minute per ton mined. The Five-foot workings, 24 acres, were producing 2.5 cubic feet of methane a minute to every 1,000 feet of coal exposed, or slightly more than 1½ cubic feet a minute per ton mined. These figures, however, do not give as close a comparison as could be desired for the amount of gas given off per square foot in working faces is greater than in most old workings. The Five-foot and Red Ash workings contained

a large proportion of fresh-coal exposures, whereas the extensive Hillman and Baltimore workings were in large part old, although in places they had many fresh working faces. The most notable features are the strong, general and local emanation of methane from the Hillman and Five-foot beds, the more moderate amount from the Red Ash bed and the very small general emanation from the extensive Baltimore workings.

HOYT MINE.

The Hoyt mine, which is one of the newer collieries of the Pennsylvania Coal Co., is in the center of the main basin 2 miles southwest of Pittston. Beds from the Pittston to the Red Ash, inclusive, are mined extensively. The workings are under the south side of the Susquehanna River for some distance and the northwest part of the mine extends under the river. North of the shaft there is a broad low anticline showing a number of crenulations to the northeastward. South of the shaft the measures slope into a shallow basin, in which the Red Ash bed is a few feet below sea level, and then rise very rapidly in a narrow steep-sided anticline, the crest of which is 238 feet higher than the bottom of the basin. This upturn is the southern margin of most of the workings, but the Red Ash bed has been followed up the rise by a gangway known as the south slope.

The mine is reported to be gaseous, especially in the workings under the river where the cover is thick and there is a sheet of clay in the valley filling. The sharp uplift south of the shaft was not as gaseous as was expected, but one heavy blower came from it in the workings in the Pittston bed. There have been several vigorous blowers in the mine. One, which continued to give off gas for many months, was from a crevice more than 16 feet deep. Another was in the roof at the end of the main gangway.

AIR SAMPLES FROM RED ASH RETURNS.

In investigating conditions in the Hoyt mine the Red Ash bed was selected because the amount of gas in its different returns varied considerably and the air currents were advantageously arranged for being sampled separately. The following table gives the results of analyses of five samples of return air, and the volume and velocity of the air at the points of sampling on November 18, 1910. The returns were also measured and sampled April 29, 1912, after mining had been suspended nearly a month. The results of the later work are given on page 145. In figure 29 are shown the extent of the workings in the Red Ash bed up to the end of 1910, the points at which air was sampled, the direction of the air currents, and the location of working faces.

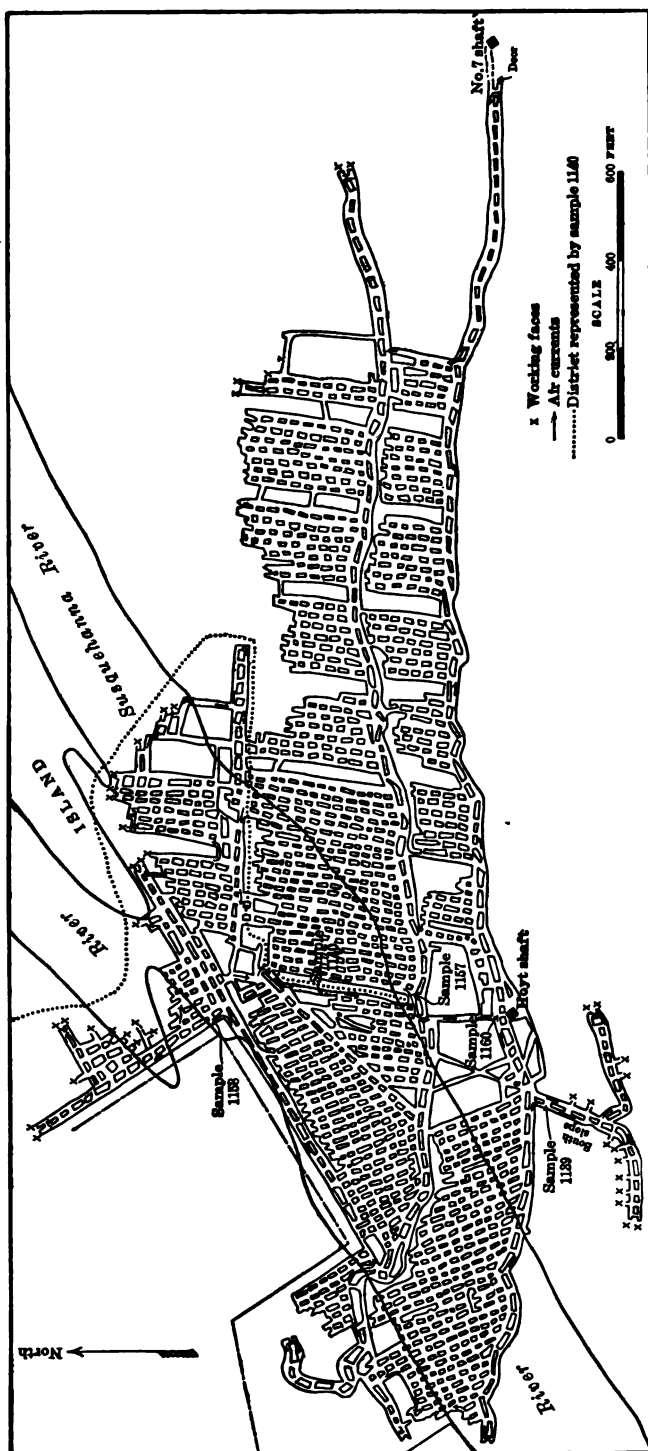


FIGURE 20.—Map of Red Ash workings in Hoyt mine, showing course of air currents sampled in November, 1910.

Analyses of return air in Red Ash workings, Hoyt colliery.

Lab- ora- tory No.	Return.	Place of sampling.	Air current.		Methane.		Carbon diox- ide.
			Vol- ume per minute.	Veloc- ity per minute.	Pro- por- tion in mine air.	Vol- ume per minute.	
1158	Red Ash.....	150 feet southwest of sta- tion 611, 106 feet above sea level.	<i>Cu. ft.</i> 22,491	<i>Feet.</i> 306	<i>Per ct.</i> 1.27	<i>Cu. ft.</i> 286	<i>Per ct.</i> 0.06
1140	Red Ash, west split.....	Return from north of sta- tion 895.	16,626	978	1.53	254	.11
1157	Red Ash, east split.....	Station 679.....	33,904	485	1.26	460	.08
1160	Red Ash, main return.....	Near shaft.....	75,680	946	1.27	961	.08
1139	Red Ash, south slope.....	210 feet southwest of shaft.	13,388	200	.94	126	.07

To ventilate the Red Ash workings in the Hoyt mine an air current of about 76,000 cubic feet a minute was used. It passed down the main shaft and was distributed through the mine in various splits. One larger split passed directly out through the east workings and then, circling to the north, drained all territory south and east of the dotted line in figure 29. The split was sampled at the place marked "sample 1157" (fig. 29), just in by the junction with other returns.

Another split ventilated the No. 1, or south slope workings, a small but gaseous district being actively mined. Beyond the point where sample 1139 was taken, the split turned west through a main airway, and finally, after dilution with about 50 per cent of fresh air, ventilated the west workings; then passing east for some distance it turned south at the place where sample 1158 was taken. Sample 1158 therefore represented the return air from No. 1 slope and that from the west workings, the ventilation of the latter being in part effected by fresh air direct from the downcast in the shaft.

Another split ventilated the small area of active workings north of the river, returned south, passed east through the workings under the river that are inclosed in the loop of the dotted line in figure 29, then west again, and finally it turned south along the west side of a narrow pillar which separates the ventilation of the east workings, and passed the point where sample 1140 was collected.

One hundred yards south of the place where sample 1140 was taken this air joins the air from the east workings in a main return which 200 feet farther south reaches the upcast at the shaft, where sample 1160 was taken.

RÉSUMÉ OF CONDITIONS IN RED ASH WORKINGS.

The analyses of the samples taken show no great variation in the amount of methane in different parts of the Red Ash workings, if the relative extent of the workings and of fresh surfaces be considered. The air in sample 1157 (see fig. 29) represents a large territory, mostly of old workings, and carries 460 cubic feet of methane a minute. The

very small area of active mining on the uplift on the south slope yielded methane at the rate of 126 cubic feet a minute. The west workings (sample 1158) yielded 160 cubic feet of methane a minute from a district that is large but was not being mined at the time of sampling.

The air from the north workings, which include an area where are many undulations of the strata, is represented by sample 1140. This return ventilates a part of the mine which was of small extent, but was being actively mined and was giving off considerable gas. It carried 254 cubic feet of methane a minute, or little less than the volume from the large area of old workings on the west side. The aggregate volume of methane in the three returns was 1,000 cubic feet a minute when measured separately and 961 cubic feet a minute in the total return at the shaft upcast.

MINE NO. 14, NEAR PITSTON.

The large No. 14 mine of the Pennsylvania Coal Co. is near the south bank of the Susquehanna River about 3 miles southwest of Pittston. The coal measures here are in a deep basin and along the river flat they are covered by beds of sand, gravel, and clay that in places aggregate more than 100 feet thick. There are several important beds of coal, but the principal workings are in the Pittston, Checker, Diamond, and Hillman beds. The mine has always been classed as gaseous and has had numerous small gas explosions.

AIR SAMPLES.

As the workings differ considerably as to depth, structure, and amount of gas given off in different places, samples of air were collected in the principal returns to detect variations that might be connected with special conditions. The places at which the samples were taken and the ventilation that some of them represent are shown in Plate VI. The samples were analyzed at the Pittsburgh laboratory of the Bureau of Mines with the following results:

Analyses of return air in No. 14 mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Methane.		Proportion of carbon dioxide in mine air.
			Volume per minute.	Velocity per minute.	Proportion in mine air.	Volume per minute.	
1153	Pittston bed, first slope....	Head of motor plane.....	<i>Cu. ft.</i> 26, 520	<i>Feet.</i> 1,020	<i>Per ct.</i> 0. 89	<i>Cu. ft.</i> 236	<i>Per ct.</i> 0. 14
1152	Pittston bed, main return....	North pitch workings.....	16, 560	238	. 72	119	. 10
1151	do.....	Rise to Marcy bed, near shaft.....	22, 360	860	. 47	105	. 11
1150	do.....	Near shaft.....	104, 728	848	a. 41	429	. 13
1154	Checker bed, main return....	Near upcast.....	147, 700	1, 692	. 26	384	. 06

a Another sample gave 0.40 per cent.

AIR SAMPLES FROM PITSTON RETURNS.

The first sample (Lab. No. 1153), taken in the return at the head of the motor plane in the first slope, where there were 25 working faces, represented an area of workings about 200 feet wide and 1,400 feet long, most of which was under the river. The return carried 236 cubic feet of methane a minute.

Sample 1152 was taken in the main return from the north pitch workings, which are nearly all under the river and Monocanock Island. The workings, which had 12 working faces, were 1,400 feet long by 200 to 450 feet wide. The return air carried 0.72 per cent methane, or 119 cubic feet a minute.

Sample 1151 was taken from an extension of the return from which sample 1152 was taken, at a rise up which the current was carried to ventilate the workings in the Marcy bed. The return gained somewhat in volume by leakage inward through doors near the shaft, so that at this place it had a volume of 22,360 cubic feet per minute and carried only 0.47 per cent of methane, or 105 cubic feet a minute.

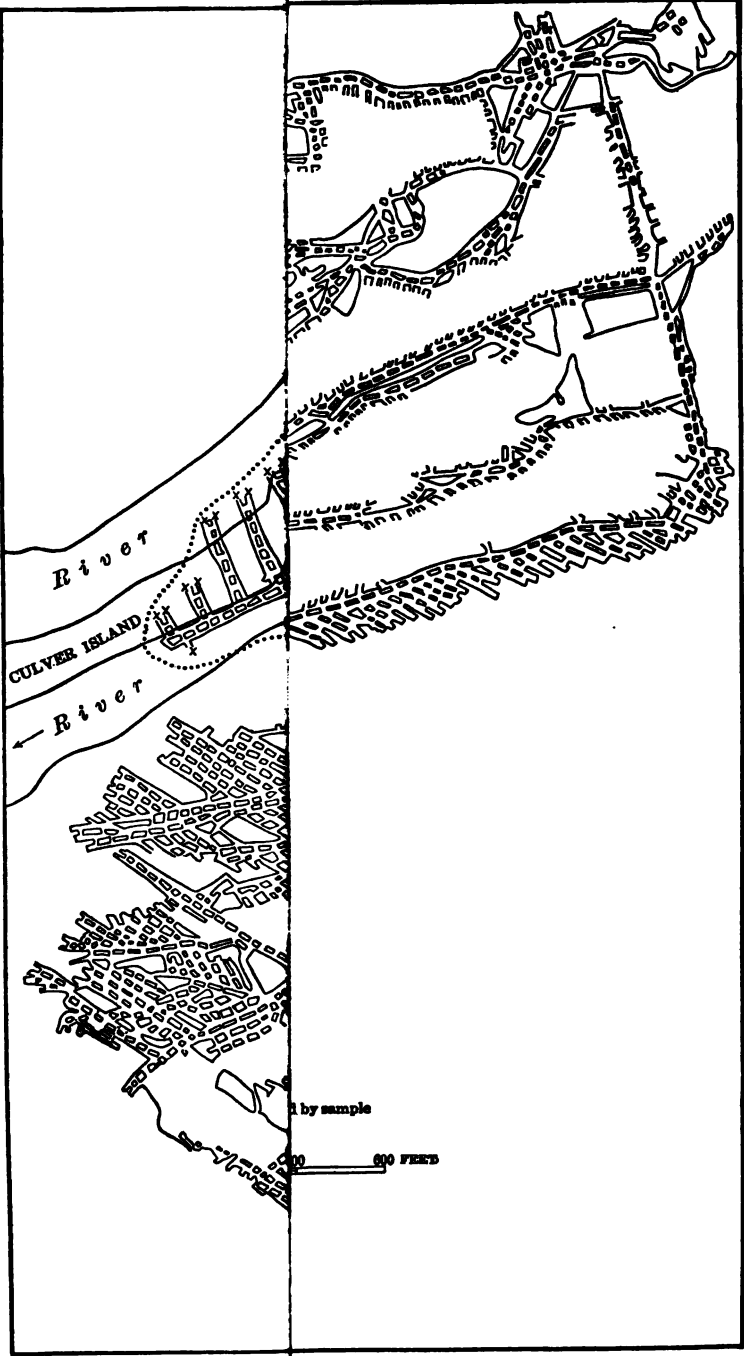
Sample 1150, taken in the main Pittston return near the shaft, represented all the return air from that bed except the part represented by sample 1151, which passed into the Marcy workings. The return air, which had a volume of 104,728 cubic feet a minute, carried 0.41 per cent of methane, so that a total of 429 cubic feet of methane a minute was passing up this upcast.

AIR SAMPLE FROM CHECKER RETURN.

A sample (Lab. No. 1154) was taken of the main return from workings in the Checker bed. This coal varies from $3\frac{1}{2}$ to 7 feet in thickness and was being worked about as extensively as the Pittston bed in the same area, with 85 working faces. The return near the shaft was passing 147,700 cubic feet of air a minute, carrying 0.26 per cent methane, or 384 cubic feet a minute. The close similarity between the volume of methane in this return and that in the main upcast from the Pittston bed is interesting. The Pittston bed is very much thicker and has more extensive old workings, a fact that indicates that the Checker bed is giving off much more methane for each square foot of coal exposed.

COMPOSITION OF PITSTON COAL.

A sample of coal from the Pittston bed in the No. 14 mine was collected November 21, 1910, from a working face near station 4695 on the Brannagan slope. It represented the entire face except a thin parting of bony coal which was rejected. The sample was analyzed in the laboratory of the Bureau of Mines at Pittsburgh with the following results.



Results of analysis of coal from Pittston bed in mine No. 14, 3 miles southwest of Pittston, Pa.

[A. C. Fieldner, analyst.]

	Condition.			Referred to coal. "Moisture and ash free."
	Air dried.	As received.	Moisture-free.	
Proximate analysis:				
Moisture.....	0.60	2.19		
Volatile matter.....	5.76	5.67	5.80	6.17
Fixed carbon.....	87.64	86.24	88.17	93.83
Ash.....	6.00	5.90	6.03	
	100.00	100.00	100.00	100.00
Ultimate analysis:				
Hydrogen.....	2.56	2.70	2.52	2.68
Carbon.....	87.77	86.37	88.30	93.97
Nitrogen.....	.92	.91	.93	.99
Oxygen.....	2.17	3.55	1.64	1.74
Sulphur.....	.58	.57	.58	.62
Ash.....	6.00	5.90	6.03	
	100.00	100.00	100.00	100.00
Calorific value determined:				
Calories.....	7,807	7,682	7,854	8,358
British thermal units.....	14,053	13,828	14,137	15,044
Calorific value calculated from ultimate analysis:				
Calories.....		7,771		
British thermal units.....		13,988		

* Loss in air drying, 1.60 per cent.

SLOAN MINE.

The Sloan mine is situated in the outskirts of Scranton, its shaft being 2 miles west of the city hall. There are extensive workings in the upper beds, but the No. 2 Dunmore or lowest bed had only recently been opened. A moderate amount of gas had been encountered in this bed and the proportion of it in the returns varied markedly in different parts of the workings. After many mines in the Lackawanna Valley had been inspected in order to ascertain the conditions under which gas occurred it was decided that the Dunmore workings at Sloan presented particularly favorable conditions for special study of the escape of gas. The area of the workings was relatively small, the air currents were so apportioned that their composition was representative of the air in the separate parts of the mine which they ventilated, and the structural conditions presented certain marked variations.

AIR SAMPLES FROM DUNMORE RETURNS.

A time was selected when atmospheric and other conditions were normal and the miners were evenly distributed through the working faces in the different parts of the mine. Two foremen perfectly familiar with the ventilation system conducted the author through the returns to points where he could collect samples representing the

gaseous emanations from definable districts in the workings. Five such samples (Lab. Nos. 1234, 1235, 1237, 1238, and 1239) were obtained and a sixth sample (Lab.No. 1236) was taken to represent the total return near the foot of the main upcast at the foot of the shaft.

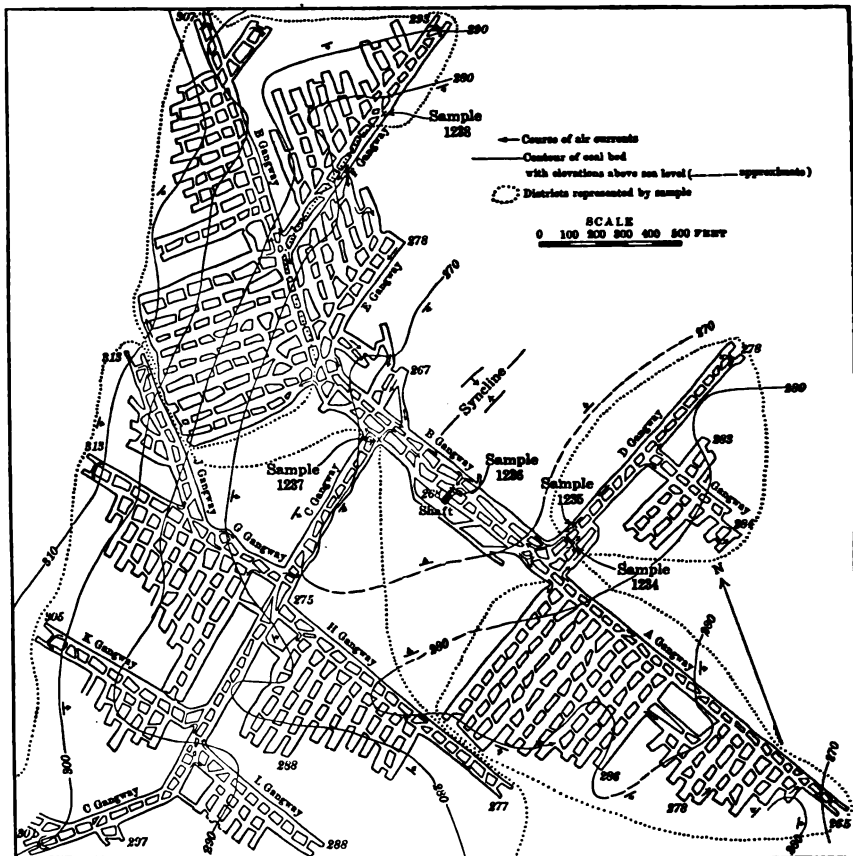


FIGURE 30.—Map of workings in Dunmore bed, Sloan mine.

Cross sections of gangways were measured and duplicate anemometer readings made at each place. The points at which samples were taken are shown in figure 30.

The samples were analyzed at the Pittsburgh laboratory of the Bureau of Mines with the following results.

Analyses of return air in Dunmore workings, Sloan mine.

[G. A. Burrell, analyst.]

Lab. No.	District in workings.	Air current.		Methane.		Carbon dioxide.
		Volume per minute.	Velocity per minute.	Proportion in mine air.	Volume per minute.	
1234	Gangway A.....	<i>Cubic feet.</i> 15,794	<i>Feet.</i> 175	<i>Per cent.</i> 1.02	<i>Cubic feet.</i> 181	<i>Per cent.</i> 0.08
1235	Gangways D and I.....	15,470	180	.65	101	.03
1237	Gangways C, G, H, J, K, and L.....	45,800	382	1.08	495	.09
1238	Gangways B and F.....	15,270	402	.08	12	.09
1239	do.....	1,065	142	.03	4	.07
1236	Total return.....	94,080	980	.70	659	

In the development of the Dunmore workings the greatest outflow of gas was found in gangways H and L (included in sample 1237) and in gangway A (sample 1234). In gangways B and F (sample 1238) no gas had been reported and it was seldom appreciable by lamp test in the headings. In gangways A, H, and L notable amounts of gas were found in all the headings, but the amount varied at different headings and at the same heading. Many "blowers" have been encountered and sometimes these have been ignited, but without causing any special damage. All the holes drilled for shots gave off more or less gas.

STRUCTURE OF DUNMORE BED.

The structure of the Dunmore bed in the workings at the Sloan mine is shown in figure 30 by contour lines with 10-foot intervals. It may be seen from this map that the strata are only slightly disturbed from their original horizontal position and have about the same average elevation above sea level. At the shaft and to the east there is a shallow syncline or basin the axis of which rises gently to the southeast in the area represented by sample 1237. That area in fact is in beds dipping gently to the eastward with a slight rise over a low anticline on gangways H and L and a steeper upgrade on gangway I. Sample 1234 represented the air from an area occupied by the eastward extension of the anticline first mentioned, but at the further end the workings cross a very shallow basin and rise over a low local uplift. Sample 1235 was from an area where the beds are nearly horizontal or dip gently north on the south side of the central syncline. Sample 1238 was taken at a point on the north side of the syncline, opposite sample 1235, where the dips are steeper. The coal on this side rises from an altitude of 280 feet along gangway F to 312 feet in the easternmost working faces off gangway B. The dip in this rise, however, is only 3°.

No notable faults or breaks were observed.

CHARACTER OF DUNMORE COAL.

The No. 2 Dunmore bed is 4 to 4½ feet thick in the gaseous workings south of gangways A, H, and L. To the north in the nongaseous region the coal is in two beds, 2 to 2½ feet of rock separating the upper bed, which is 2 feet thick, from the lower bed, which is 1 foot thick. The gaseous coal is dry and hard and contains considerable dirt, whereas the nongaseous coal has a much larger moisture content and is softer.

A representative sample of coal from the most gaseous part of the Dunmore No. 2 bed in the Sloan mine was taken from the heading in gangway H on December 5, 1910, and analyzed in January, 1911, in the laboratory of the Bureau of Mines at Pittsburgh with the following results:

Results of analysis of coal from Dunmore No. 2 bed, Sloan mine, near Scranton, Pa.

[A. C. Fieldner, analyst.]

	Condition.			Referred to coal "Moisture and ash free."
	Air dried.	As received.	Moisture free.	
Proximate analysis:				
Moisture.....	0.95	3.43		
Volatile matter.....	6.96	6.79	7.03	7.96
Fixed carbon.....	80.26	78.25	81.03	92.02
Ash.....	11.83	11.53	11.94	
	100.00	100.00	100.00	100.00
Ultimate analysis:				
Hydrogen.....	2.30	2.52	2.22	2.52
Carbon.....	80.87	78.86	81.65	92.72
Nitrogen.....	.79	.77	.80	.91
Oxygen.....	3.74	5.87	2.91	3.30
Sulphur.....	.47	.46	.48	.55
Ash.....	11.83	11.53	11.94	
	100.00	100.00	100.00	100.00
Calorific value determined:				
Calories.....	7,263	7,101	7,353	8,350
British thermal units.....	13,109	12,782	13,235	15,030
Calorific value calculated from ultimate analysis:				
Calories.....		6,998		
British thermal units.....		12,596		

a Loss in air drying, 2.48 per cent.

AMOUNT OF METHANE IN RELATION TO AREA OF COAL EXPOSED.

The chambers and gangways shown in figure 30 were measured with a chartometer to ascertain the total length of all excavations in each district represented by samples 1234, 1235, 1237, 1238, and 1236. Multiplying the length of excavation by the average height of the coal exposed, 4½ feet in districts south and 3 feet in districts north of the shaft, gives the approximate number of square feet of coal exposed.

Volume of methane relative to area of coal exposed, Sloan mine.

Lab- oratory number of sam- ple.	Return.	Area of coal exposed.	Volume of methane per minute per 1,000 square feet of coal exposed.	Area of coal ex- posed to each cubic foot of methane.
1238	Gangways B and F.....	<i>Sq. feet.</i> 81,900	<i>Cu. feet.</i> 0.15	<i>Sq. feet.</i> 6,825
1237	Gangways C, G, H, J, K, and L.....	151,300	3.28	305
1235	Gangways D and I.....	30,175	3.37	299
1234	Gangway A.....	87,975	1.81	547
	Region between samples 1238 and 1236.....	35,100		
1236	Total return.....	386,450	1.70	586

The most noteworthy feature shown in this table is that the relatively large area of coal in the district represented by sample 1238 was giving off remarkably little methane. Sample 1237 represented air containing gas from a large amount of coal in pillars, chambers, and gangways, so that the amount of methane per square foot was less than for the small area in the district represented by sample 1235, which, however, gave off only 101 cubic feet of methane a minute, whereas the district represented by sample 1237 was giving off 495 cubic feet. This indicates that a large part of the workings drained by the return in which sample 1237 was taken was not especially gaseous, the large emanations coming from headings and new chambers. A sample (Lab. No. 1239) of the return air in gangway E representing the short interval of gangways and chambers between gangway F (sample 1238) and the main upcast (sample 1236) near the shaft had practically the same methane content as sample 1238.

CARBON DIOXIDE IN MINE AIR.

In four samples the carbon dioxide content was determined. The average proportion in the mine air was about 0.05 to 0.08 per cent in excess of that in ordinary atmospheric air. With the volume of the upcast air 94,080 cubic feet a minute, this 0.08 per cent amounts to only 75½ cubic feet of carbon dioxide a minute, or not much more than one-tenth as much as the volume of methane. Doubtless this is not much more than should be expected from blasts, lights, the breath of men and mules, and various minor sources.

CHANCE OF LEAKAGE OF GAS TO HIGHER WORKINGS.

Above the Dunmore workings in the Sloan mine is an extensive mined area in the next coal bed known as the Clark. Between the two beds are sandstone and clay beds 163 feet thick which seemingly were so nearly impervious that little gas could escape through them. Moreover, the strata are so regular that any possible leakage would

probably be uniform throughout the area. Some years ago a considerable flow of gas was found coming up through a crack in the floor of the Clark workings, presumably from the Dunmore coal below. The crack was cemented and a pipe set into it to conduct the gas to the main upcast.

EFFECT OF CESSATION OF MINING ON MINE AIR.

RESULTS OF SAMPLING AIR OF IDLE MINES.

During April, 1912, the mines in the anthracite field were closed because of a strike. Advantage was taken of this circumstance to collect air from several of the mines at the same localities as in previous visits in order to make a comparison between the outflow of methane during mining and that at a time when there had been no mining for a month previous. In most mines the volume of ventilating air was greatly reduced during the period of no mining, so that if much of the methane normally in the return air is given off by the great area of ribs and pillars the percentage of methane would be expected to increase in the returns. This proved to be the case in some instances, but in most of the returns the volume of methane had diminished considerably and with many unexpected variations. In a few workings there had been some slight changes in arrangement of the air currents so that exact comparison could not be made. The samples collected were analyzed in the laboratory of the Bureau of Mines at Pittsburgh. A comparison of the results with those obtained from the samples collected in the same place in December, 1910, and January, 1911, is presented in the table following.

Methane in return air during mining (November, 1910, to January, 1911) and after mines had been shut down nearly a month (April, 1912).

[G. A. Burrell, analyst.]

Mine.	Return.	November, 1910, to January, 1911.		April, 1912.	
		Proportion of methane in mine air.	Volume per minute.	Proportion of methane in mine air.	Volume per minute.
		<i>Per cent.</i>	<i>Cubic feet.</i>	<i>Per cent.</i>	<i>Cubic feet.</i>
Lance.....	Bennett and overlying beds, total up- cast, No. 1 air shaft.....	0.96	a 1,920		
Do.....	Red Ash bed, total upcast, No. 2 air shaft.....	1.07	1,067		
Do.....	Ross, total upcast, No. 5 slope.....	1.46	714	b 2.29	1,375
Do.....	Total upcast.....		c 2,691	1.56	a 3,120
Do.....	Cooper, west, No. 11 tunnel.....	2.68	508	2.98	678
Do.....	Cooper, east, No. 23 tunnel.....	1.04	431	1.55	471
Do.....	Five-foot, east, at No. 7 slope.....	1.46	383	d 3.06	d 416
Do.....	Five-foot, west, at No. 7 slope.....	.23	65	e 2.20	e 523
Do.....	Hillman, east, No. 14 tunnel.....	1.50	555	2.64	706
Do.....	Hillman, west, No. 14 tunnel.....	.55	167	f 1.56	f 147
South Wilkes- Barre (No. 5).	Total upcast of all beds.....	.72	2,269	.71	2,231
Do.....	West Hillman returns.....	1.43	432	2.96	611
Do.....	Abbott workings north of shaft 5, plus Stanton No. 5 return and Hillman No. 4 tunnel.....	.67 .43	124 501		
Do.....	Total Abbott workings south of shaft.....	1.14	230	.55	51
Dorrance.....	Return from Red Ash.....	A .99	865	1.03	927
Do.....	do.....	1.07	1,024		
Do.....	do.....	1.34	1,080	.47	973
Do.....	Baltimore return, from Baltimore.....	.49	968		
Do.....	Red Ash and Five-foot beds, north side.....	.73	1,161	.77	1,375
Do.....	Baltimore return, from Baltimore, Red Ash and Five-foot beds, south side.....	.64 1.02	652 1,427		
Do.....	Five-foot return (§).....	.84 1.38	157 191	1.16	299
Do.....	Hillman returns, 2 splits.....	A 92 to 112 f .55 to .64	1,777 720		
Hoyt.....	Red Ash return, 150 feet southwest of station 611.....	f 1.27	286	1.53	229
Do.....	Red Ash, return from No. 1 slope.....	f .94	126	1.29	80
Do.....	Total return, Red Ash.....	f 1.27	961	1.53	475
Sloan.....	Dunmore workings, gangway A.....	A 1.02	161	.51	147
Do.....	Dunmore, gangways D and I.....	A .65	101	.31	77
Do.....	Dunmore, gangways C, G, H, J, K, and L.....	A 1.08	495	.76	296
Do.....	Dunmore, gangways B and F.....	A .08	12	.0	0
Do.....	Dunmore, total return.....	A .70	659	.49	464

a Air volume, 200,000 cubic feet a minute.

b No. 3 slope, total Red Ash and Ross returns.

c Sum of three preceding figures.

d Not worked for several months. Ventilated by split from Ross-Red Ash air, the methane of which is not subtracted.

e Now on split of air from Ross-Red Ash workings, the methane introduced by that air not subtracted.

f Ventilated by air from split of Ross-Red Ash air, carrying 1.27 per cent methane, or 217½ cubic feet a minute.

g Total result, less 217½ cubic feet from Ross-Red Ash split.

A December, 1910.

f Sampled by H. I. Smith, early in April, 1912.

f November 19, 1910.

DISCUSSION OF RESULTS.

These figures indicate that with the cessation of mining the emanation of methane diminished in some places whereas in others it did not. In some of the workings the volume of methane in samples taken after four weeks of cessation was greater than when the sampling was done in December, 1910, or in January, 1911. Of course the comparisons would have more value if the first set of samples had been taken immediately before mining ceased, for doubtless there had been some changes in the volume of gas given off during mining after the earlier sampling.

LANCE MINE.

There was an increase of nearly 20 per cent in the methane in the total upcast air in the Lance mine, if it be assumed that fan No. 2 was moving 200,000 cubic feet of air a minute on both occasions. It happened that at this fan the volume could not be measured satisfactorily. Only this one fan was working during the shut down and therefore the volume of air coming out of the mine was considerably diminished. At this time also the Red Ash and Ross air came to this fan, after being used en route for ventilating workings that formerly had been on fresh-air splits. Increased methane emanation is shown in the Cooper and east Hillman returns, the Hillman workings being ventilated by a split of Ross-Red Ash air at the time the second sample was taken. Both the east and west workings in the Five-foot bed were also ventilated by a split of Ross-Red Ash air, but the proportion of methane in this split was not tested at the intake on April 27, so that the increment of methane that this air received in passing through the Five-foot workings is not known.

SOUTH WILKES-BARRE MINE.

The cessation of mining appears to have had slight effect on methane emanation in the South Wilkes-Barre mine except perhaps to cause a marked diminution in the volume from the south workings in the Abbott bed. The north workings in the Abbott and Hillman beds show material increase. The fan was running at the same speed at each time of sampling, 45 revolutions per minute. The volume of air delivered, which was measured in December, 1910, was approximately 315,083 cubic feet a minute.

DORRANCE MINE.

The Dorrance mine shows frequent variations in methane emanation and the volumes found on April 27 do not show much diminution due to the cessation of mining. The volume furthermore in the total Hillman return was less than when tested 1½ years before, but greater than when sampled by H. I. Smith the month before.

In the return from the south half of the Baltimore workings there was slight increase in methane content; in the Five-foot returns, a decided increase; and in the Red Ash returns, a material increase.

HOYT MINE.

The volume of methane in all the Red Ash returns in the Hoyt mine decreased after mining had ceased, the total volume diminishing 50 per cent, and that in the return from No. 1 slope, 30 per cent.

SLOAN MINE.

In the Dunmore workings in the Sloan mine there was a marked decrease in the amount of methane during the four weeks of no mining, the total volume dropping from 619 to 404 cubic feet a minute, the amount in all of the gangways lessened correspondingly, the emanation diminishing to 60 per cent in the very gaseous C, G, H, J, K, and L gangways, and to zero in the B and F gangways, which had been only slightly gaseous.

RÉSUMÉ.

The causes of variations in the methane emanation during the period of no mining are not at all clear. On the whole there was a marked decrease, which was to be expected, because less fresh coal was being mined and the gangways and workings were not being advanced into undrained territory, but on the other hand, the results show that some districts continue to give off the same amount of gas as when mining was active, and in places there was a marked increase. It was not possible to connect these places with differences in structural features (anticlines, synclines, or faults) nor to show that the greater escape of gas was due to crushing or creeping, or to differences in composition or texture of roof, floor, or adjoining beds, all of which may in certain places be factors of some importance.

MISCELLANEOUS OBSERVATIONS IN MINES IN THE ANTHRACITE FIELD.

MINES IN AND ABOUT SCRANTON.

Many of the mines about Scranton were inspected in the course of the investigation. The mines are extensive and include vast areas of old workings which are not especially gaseous. Gas in this district appears mostly in the working faces and occurs very irregularly. In general, the total volume of methane is not nearly so large as in the Pittston-Wilkes-Barre district. The smaller volume is due partly to the proximity of the coal to the surface and partly to the extent of the old workings, which have drained off part of the gas. Usually the largest volume and most noticeable manifestations of gas are in the Dunmore beds near the base of the measures.

BELLEVUE AND PANCOAST MINES.

In the lower part of the Bellevue mine in Scranton the working faces of the Dunmore bed are very gaseous in places.

In the Pancoast mine at Throop the fresh workings in the Dunmore No. 1 bed are especially gaseous, and blowers occur at intervals. The fresh coal gives off so much methane that the gas can be ignited by placing a light between the lumps in a car, and blasting often lights gas in crevices in the solid, so that water service for extinguishing such fires has to be maintained at the working breasts. The author noted several vigorous blowers in workings in this mine under the Lackawanna River and on the slopes south. The measures dip gently to the northwest. At one point there is a sharp synclinal crumple in the lower beds, on one side of which there is much more gas than on the other side, but no reason for the difference was apparent. Bore holes sunk to the Dunmore bed from the Clark workings emitted a large volume of gas and one of them also had a flow of water.

MARVINE MINE.

At the Marvine mine, just north of Scranton, considerable gas was being given off in the Dunmore workings, the amount increasing in slips and breaks and in the workings under the river. Some years ago a strong blower was opened in the workings of the next bed above (Clark). The gas, which was under great pressure, was finally controlled and utilized by cementing a pipe into the rock and leading the gas to the surface. The flow continued for a long time.

DICKSON MINE.

In the Dickson mine in Scranton the conditions are similar to those in the Marvine mine, as the Dunmore bed gives off much gas in the fresh workings. Several roof feeders were opened, but they have been finally exhausted. The workings extend to the bottom of the syncline under the river, but the amount of gas appeared to be less in the deeper chambers than in districts where the beds are locally arched or broken.

GREEN RIDGE AND DIAMOND MINES.

In the Green Ridge mine, in the northern part of the city of Scranton, the coal is much nearer the surface and gives off less methane, except in a few places in the Dunmore No. 2 bed.

In the Diamond mine, near the center of the city, the Dunmore beds are about 450 feet below the surface in the bottom of the basin. Tests of gas in returns from No. 1 and No. 2 Dunmore beds and the New County bed on February 8 to February 10, 1910, made by the Delaware, Lackawanna & Western Coal Co. showed 0.2 and 0.3 per

cent of methane except in the Riverside return, No. 1 Dunmore, where 0.6 per cent was found. The total volume of the air currents measured was 95,600 feet per minute and the total volume of methane in them was 246 cubic feet a minute, which is very little in comparison with some returns in mines near Wilkes-Barre.

STORRS, CAYUGA, AND BRISBIN MINES.

The extensive workings of the Storrs mine are largely in the rise toward the northern outcrop of the beds on the slopes north of Providence. The conditions governing the occurrence of gas in this mine are fairly typical of wide areas along the north margin of the basin of the Scranton region. The results of numerous tests of the gas in the returns from the Dunmore, Clark, Pittston, Diamond, and other beds were given to the author by the Delaware, Lackawanna & Western Coal Co. The volume of the air currents tested ranged from 7,200 to 45,000 cubic feet a minute. In most of the returns there was no appreciable methane or but a trace. At a few places the methane content was 0.1 to 0.2 per cent, and in one restricted area the content in a current of 19,125 feet was 1.1 per cent.

Similar conditions were found in the Cayuga mine, where it is reported that samples taken in February, 1910, from returns from the Rock bed, the Four-foot bed near the shaft, and at the head of the Dunmore slope show no trace of methane. Tests made by the company at the same time in the Brisbin mine showed only traces and up to 0.1 per cent of methane in returns from the Dunmore, Clark, Diamond, and Rock beds. One return of 40,000 cubic feet from extensive old workings in the Clark bed carried only 0.1 per cent, or 40 cubic feet of methane a minute.

MANVILLE AND GRASSY ISLAND MINES.

In the Manville mine in Scranton the workings are in the south slope of the basin. The results of some tests of returns made in February, 1910, have been reported to the author by the Delaware, Lackawanna & Western Coal Co. The Dunmore returns carried methane from a trace to 0.2 per cent except in the east split return from No. 2 Dunmore where there was no trace of methane. It would appear from the analyses that the total emanation of methane from the Dunmore workings of this mine was somewhat less than 200 cubic feet a minute. The Dunmore return air, near fan No. 1, carried 0.1 per cent of methane in a current of 93,600 cubic feet a minute, or nearly 194 cubic feet of methane a minute.

A remarkable sporadic area of increased methane emanation was observed at Grassy Island No. 2 shaft, near Olyphant. At this place, the Dunmore bed No. 4, which is about 6 feet thick and has a heavy layer of sandstone over it, had been notably gaseous and the roof and

floor had numerous small feeders in them. Bed No. 3 was also gaseous but was being worked only to a small extent. The measures are very regular and there seems to be no explanation for the amount of methane in this mine being much greater than in those adjoining, other than some variation in the character of the coal itself.

MINES BETWEEN PITTSTON AND HUDSON.

Beside the detailed investigations in the Lance, Dorrance, South Wilkes-Barre, Hoyt, and No. 14 mines, many data were obtained as to some of the conditions in various mines in the region between Pittston and Wilkes-Barre. Several mines were also visited by H. I. Smith in the spring of 1912 for the purpose of collecting samples of return air, which were analyzed at the Pittsburgh laboratory of the Bureau of Mines. Several analyses of air from the Pettebone mine were kindly supplied by the Delaware, Lackawanna & Western Coal Co.

MINE NO. 6.

Parts of mine No. 6 of the Pennsylvania Coal Co., near Inkerman, were decidedly gaseous. There are many small abrupt flexures in the measures in this mine, but the occurrence of gas seems to bear no relation to them, as most of the gas is given off in certain zones of solid coal that show no signs of disturbance. The Clark bed has several strongly gaseous areas. In one place, where there was some local faulting of the beds and the coal was considerably crushed, there were several small blowers in the floor of the gangway. Although the blowers were conspicuous because of the noisy bubbling, the amount of gas produced was probably much less than in some of the fresh workings. Another blower in the Marcy workings, at a place where the beds suddenly pitch up a few feet, was seemingly coming from a crack in the bottom or side of the small syncline so formed.

PINE RIDGE MINE.

The Pine Ridge mine of the Delaware & Hudson Coal Co. northeast of Wilkes-Barre has been in operation many years and is still producing a large tonnage. All of the usual beds are being worked, and the lowest beds are 500 to 700 feet below the surface in a large part of the area. The principal structural feature is a syncline, but there are many subsidiary flexures, some of them sharp enough to be called faults. Some parts of the mine were gaseous and certain places gave off a very large amount of methane. An instance is on record ^a of a flow of gas from the roof in a small syncline which required an air current of 15,000 cubic feet a minute to keep the proportion of methane below the explosion limit. At one time the mixture ignited

^a Anon., Fire damp in the anthracite mines: Eng. and Min. Jour., vol. 17, 1874, pp. 33-34.

for 150 feet along the roof, although 25,000 cubic feet of air per minute was passing.

Several large feeders occurred in bottoms of synclines, especially in two with a low arch between them. It was noted in this vicinity that an underlying bed may be less gaseous, under the same structural conditions, than the bed above. One gaseous belt was in an anticline in which the coal rose to the cap of alluvium and considerable gas was found in places above water level even with only 20 to 30 feet of cover. At the time of the author's visit there was much gas in all the lower beds and workings, especially in the Ross bed where there was an old blower. This blower came from near the crest of an anticline, and a bore hole on the crest tapped a large volume of gas from the Red Ash bed some distance below. In years past much gas was found at intervals even in the higher beds along some of the "faults" mainly in the form of blowers or rapid gas emanations, which caused several explosions. Some bore holes in this area tapped large volumes of gas and in two of them gas has been flowing out of the pipe at the surface for many years.

Several samples of return air in the Pine Ridge mine were taken by H. I. Smith on March 16, 1912, and were analyzed with the following results:

Results of analyses of air samples from returns in Pine Ridge mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in mine air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
			<i>Cu. ft.</i>	<i>Fect.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Cu. ft.</i>
2427	Red Ash.....	No. 17 plane.....	32,000	368	0.17	0.07	-----	54
2398	Cooper, Bennett, and Checker.	Total return in air tunnel Cooper to Rock.	110,800	1,273	.16	.13	-----	177
2525	Hillman.....	No. 15 slope.....	16,300	354	.05	.08	20.33	80
2527	do.....	Second east off No. 13 slope, 2,500 feet from air shaft.	7,500	257	.04	.00	20.68	3
2528	do.....	do. ^a	-----	-----	.02	.02	20.74	2
2521	Kidney.....	Tunnel Kidney to Hillman.	18,800	348	.00	.07	20.67	0

^a Duplicate of No. 2527.

Sample 2427 was taken from a split of the Red Ash air that ventilated the northernmost workings, which were 600 to 620 feet below the surface and had 30 miners. The Cooper-Bennett-Checker return drains workings in these beds in about half of the mine; it carried a very moderate proportion of methane. The Hillman return, taken in No. 15 slope, represents a small area of deep-lying workings in which 20 men were mining a 7-foot bed of coal. The Kidney return was from an area about a half mile square lying 120 to 280 feet

below the surface and nearly all old workings. These two samples had a remarkably small methane content. A sample taken in the Red Ash workings of gas that had accumulated near the face while the brattice in the west gangway on No. 22 slope was being changed carried 18.5 per cent methane, 0.2 per cent carbon dioxide, and 16.6 per cent nitrogen. A second sample collected under similar conditions in the Ross bed just above this place carried 17.3 per cent methane, 0.10 per cent carbon dioxide, and 17.1 per cent oxygen.

DELAWARE MINE.

The Delaware mine of the Delaware & Hudson Coal Co. is in Hudson, a short distance northeast of the Pine Ridge mine. The conditions in the two mines were closely similar in most respects, but in the Delaware the coal beds are at shallower depths and are less irregular in structure. The amounts of gas varied irregularly in different parts of the mine, but some of the beds were very gaseous at certain places. Some of the returns in this mine were sampled by H. I. Smith on March 20 and 21, 1912, and were analyzed with the following results:

Results of analyses of return-air samples collected in March, 1912, from Delaware mine.

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in mine air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
2477	Ross.....	No. 2 gangway in No. 4 slope, Cooper return.	<i>Cu. ft.</i> 48,500	<i>Feet.</i> 570	<i>Per ct.</i> 0.26		<i>Per ct.</i> 20.18	<i>Cu. ft.</i> 78
2475	Ross and Checker.	No. 16 tunnel, in Cooper bed.	31,000	500	.24		20.65	74
2476	do.....	do.....			.23	0.09	20.51	71
2471	Ross and Bennett.	Ross return, 400 feet from shaft.	25,000	417	.17	.06	20.81	42
2472	do.....	do.....			.19	.07	20.31	47
2480	Bennett and Cooper.	No. 5 road, 100 feet from shaft.	35,500	612		.03	20.33	0
2473	Five-foot, Cooper, Bennett, and Checker.	East gangway in Cooper bed, 500 feet from shaft.	46,000	766	.07	.10	20.74	32
2474	do.....	do.....			.05	.07	20.67	23

^a Duplicate of preceding sample.

Most of the returns given above are splits which drain extensive workings. The Ross return, from which sample 2477 was taken, came from an area about one-half mile square, most of it lying 200 feet below the surface, and the bed is only 30 inches thick. The return (samples 2475 and 2476) from the Ross and old Checker workings drained about 30 acres in the Ross and 27 acres in the Checker. The air represented by sample 2471 drained the Ross workings 1 mile long by 400 to 1,000 feet wide, in part old workings, and also part of the Bennett workings, and carried little gas considering the great

area of coal surface that is ventilated. The Bennett-Cooper return drained extensive workings, mostly old ones. The Bennett bed is about 140 feet and the Cooper 160 feet below the surface, and the absence of gas in the return from this large area of coal is difficult to explain. Samples 2473 and 2474 were taken from a return from a large area of the Cooper workings and a small area of recent Five-foot and Checker workings in greater part not very far below the surface. A sample of air from a face in the Ross workings, on a gangway known as " $\frac{1}{2}$ west," off No. 6 slope, contained 33.4 per cent methane, 0.10 per cent carbon dioxide, and 13.25 per cent oxygen.

MINES SOUTH OF WILKES-BARRE.

BALTIMORE NO. 5 MINE.

The Baltimore No. 5 mine of the Delaware & Hudson Coal Co. is near the southeastern margin of the city of Wilkes-Barre. The strata are in the long southeastern rise out of the deep central basin and present moderate dips to the north with a few local interruptions. All the beds are worked and the depth of the lowest one, the Red Ash, is 800 feet and more in places. In general, this bed gave off more gas than the others, but the variations seemingly were not related to the structure except that the emanation increased with depth. One feeder of considerable volume was observed in the roof in the Red Ash workings. There was a large influx of water in the floor below it. Several returns in this mine were sampled by H. I. Smith on March 20, 1912, which were analyzed with the following results:

Analyses of return-air samples collected in March, 1912, from Baltimore No. 5 mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in mine air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
			<i>Cu. ft.</i>	<i>Fect.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Cu. ft.</i>
2465	Red Ash.....	East side return 75 feet from shaft.	83,200	440	0.08	0.07	20.63	66
2466	do.....	do. a			.07	.07	20.73	56
2467	Baltimore.....	First east gangway.....	27,300	224	.60	.06	20.68	164
2468	do.....	do. a			.56	.07	20.71	153
2469	do.....	West side main return...	69,500	662	.35	.06	20.67	243
2470	do.....	do. a			.36	.05	20.69	249

a Duplicate of preceding sample.

The Red Ash return current was a single split which drained about 150 acres, in which about 120 miners were at work. The bed is 9 to 12 feet thick and although considerable amounts of gas were given off in places the total volume was small. The two Baltimore return

air currents were splits off the main east and west return currents, part of which drained extensive old workings. The total volume of the Baltimore return air was stated to be 120,000 cubic feet a minute, and if this air has an average content of 0.40 per cent of methane the total emanation from that bed was 480 cubic feet a minute.

CUNNYNGHAM MINE.

The Cunnyngham mine of the Delaware & Hudson Coal Co. is in the northeastern part of Wilkes-Barre. Its chambers extend deeper into the basin than the Baltimore No. 5 workings, and some of the beds have been worked to the bottom of the basin. The mine is moderately gaseous and in it the Hillman bed begins to carry the increased amount of gas notable in this bed in a large part of the Wilkes-Barre area. The conditions in this mine were not investigated in detail, but a few samples of returns collected by H. I. Smith, March 20, 1912, were analyzed with the following results:

Analyses of return air collected in March, 1912, from Cunnyngham mine.

[G. E. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in mine air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
2461	Baltimore, east...	50 feet east of air shaft...	<i>Cu. ft.</i> 47,000	<i>Feet.</i> 470	<i>Per ct.</i> 0.11	<i>Per ct.</i> 0.03	<i>Per ct.</i> 20.62	<i>Cu. ft.</i> 52
2462	Do.....	do. a.....			.11	.03	20.62	52
2463	Baltimore, west...	Air bridge over old east gangway.	42,300	445	.16	.05	20.42	68
2459	Kidney - Abbott-Hillman.	On Hillman, 500 feet from shaft.	37,900	557	.07	.07	20.62	26
2460	Do.....	do. a.....			.10	.05	20.67	38

^a Duplicate of preceding sample.

The splits represented in the above samples drained various parts of the workings and the analyses show a remarkably uniform methane emanation. The total Baltimore return carried about 152,000 cubic feet of air per minute, and with an average methane content of 0.14 per cent the volume of methane would be about 213 cubic feet a minute, which is very small in view of the large extent and the depth of the workings. The Kidney-Abbott-Hillman return drains an area about a half mile square, in which the Hillman bed lies about 120 feet below the surface.

HOLLENBACK MINE.

The Hollenback mine, which is one of a group owned by the Lehigh & Wilkes-Barre Coal Co., is not far east of the center of the city of Wilkes-Barre. The workings extend under a district in which the strata are elevated and depressed in a marked anticline and syncline

and present numerous minor rolls and steep dips or "faults." Several beds are worked. Considerable gas is given off in places in the deeper workings, but the rate of emanation is very irregular and the variations can not be connected with the structure. Some of the faulted zones were gaseous, but this may simply indicate that the store of gas is given off more rapidly where the coal is shattered, the aggregate amount being no greater than elsewhere.

An extensive series of samples of return air in the mine were taken by H. I. Smith in March, 1912, and the results of the tests are given in the following table:

Analyses of return-air samples collected in March, 1912, from Hollenback mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in mine air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
2380	Top Red Ash.....	No. 4 slope, 75 feet from air shaft.	<i>Cu. ft.</i> 29,300	<i>Feet.</i> 381	<i>Per ct.</i> 0.39	<i>Per ct.</i> 0.07	<i>Per ct.</i> 20.61	<i>Cu. ft.</i> 114
2381	Do.....	do. ^a			.41	.08	20.64	120
2378	Red Ash-Ross....	50 feet west of air shaft..	79,200	727	.65	.07	20.31	515
2379	Do.....	do. ^a			.59	.06		477
2377	Do.....	South of air shaft.....	37,000	475	.66	.10	20.51	244
2382	Do.....	50 feet from air shaft.....	164,300	990	.26	.07		427
2383	Do.....	do. ^a			.30	.10		493
2372	Baltimore.....	East of Baltimore air shaft.	35,200	558	.20		20.04	70
2373	Do.....	do. ^a			.17	.20	19.93	60
2366	Stanton-Hillman..	At air shaft, Stanton....	63,800	1,100	.22	.08	20.49	140
2367	Do.....	do. ^a			.21	.06	20.44	134
2371	Stanton.....	No. 15 tunnel, 400 feet from slope.	33,500	782	.15	.09	20.28	50
2374	Hillman-Kidney..	Foot of Baltimore air shaft.	60,200		.14	.06	20.50	84
2375	Do.....	do. ^a			.27	.03	20.61	162
2368	Do.....	No. 22 tunnel, 700 feet from air shaft.	58,700	725	.07	.09	20.39	41
2369	Do.....	do. ^a			.09	.10	20.58	53

^a Duplicate of preceding sample.

These samples of various returns in the Hollenback mine cover a moderately large proportion of the ventilation. The amount of gas given off varied in different beds and in different parts of the same bed without corresponding variations in structure or any other observed features, although the results show that there must have been considerable diversity in the conditions governing the occurrence of the gas. The Red Ash return (samples 2380, 2381) drained an area east of the hoisting shaft. Samples 2378 and 2379 represent the return from the west side of the Ross workings and a small part of the Red Ash workings, but most of the gas was from the Ross. Sample 2377 is air from the top Red Ash from tunnels 28, 12, and 9, with some east side Ross air. Sample 2383 represents all of the return from the east side workings in the Red Ash bed, which is 14 feet thick, and is moderately gaseous in places. These samples (Lab. Nos. 2377 to

2883) probably represent the total return air from the Red Ash and Ross workings, amounting in all to a volume of 1,262 to 1,372 cubic feet of methane a minute. The relations of these beds vary greatly from place to place, but in general the Ross bed is about 9 feet thick and underlain by 18 to 35 feet of "slate". The top Red Ash is 7 feet thick and the bottom Red Ash 14 feet, with 30 to 35 feet of rock between.

The return air from the Baltimore bed was mostly from old workings of moderate extent. The Stanton-Hillman return was from old workings in which the beds dip west and, together with the return from which sample 2371 was taken, represents all the air from the Stanton workings. The Hillman-Kidney return (samples 2374 and 2375) was from the workings in these beds on the lower slopes, which extend down No. 3 plane, and samples 2368 to 2369 represent the returns from upper workings in the Hillman and Kidney beds.

Some face samples were also obtained in the Ross and Kidney workings. One, which was taken in No. 36 chamber, Ross outlet, 30 feet from the face, after an explosion in which two men were fatally burned, contained 4.12 per cent methane, 1.93 per cent carbon dioxide, and 15.78 per cent oxygen. This place had formerly shown little gas. A sample was taken in No. 48 breast off No. 13 tunnel east in the Kidney bed, 40 feet from the face, where gas had accumulated overnight. The sample contained 13.2 per cent methane, 0.4 per cent carbon dioxide, and 15.1 per cent oxygen.

MAXWELL NO. 20 MINE.

The Maxwell No. 20 mine of the Lehigh & Wilkes-Barre Coal Co. was visited by H. I. Smith in March, 1912, and the return-air sampled at various places. The results of analyses were as follows:

Results of analyses of return-air samples collected in March, 1912, from Maxwell No. 20 mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Volume of air current per minute.	Proportion of gases in mine air.			Volume of methane per minute.
				Methane.	Carbon dioxide.	Oxygen.	
			Cu. ft.	P. ct.	P. ct.	P. ct.	Cu. ft.
2529	Red Ash.....	West return, 50 feet from south shaft.	104,000	0.65	0.04	20.63	676
2402	Ross.....	No. 10 tunnel.....	18,400	.76	.04		294
2401	do.....	West airway, No. 10 tunnel.....	35,400	.75	.07	19.39	265
2404	Baltimore.....	Baltimore airway, No. 10 tunnel.....	24,400	.97	.06	20.32	237
2405	do.....	do.....		.95	.06		231
2408	Hillman.....	No. 14 tunnel to Baltimore.....	17,900	.90	.01		143
2409	do.....	do.....		.87	.10	20.58	156
2406	Abbott-Kidney..	No. 5 slope, west Hillman, last cross-cut in chamber.	29,800	1.22	1.34	18.74	364

a Duplicate of preceding sample.

These scattered samples were taken in parts of the mine that were especially gaseous. The Red Ash return was a split from the north-western corner of the Red Ash workings where about 25 miners were working. The bed is 10 to 12 feet thick and about 800 feet below sea level, or 1,400 feet below the surface. Many of the working faces liberate gas rapidly, and at times the mining has had to be suspended to let the gas drain off. The Ross return was from the deep workings to the north in which considerable amounts of gas were given off. A sample, representing air from four chambers, taken in No. 34 breast off No. 18 tunnel from a return having a volume of 115,200 cubic feet a minute, had a content of 37.40 per cent methane, 0.39 per cent carbon dioxide, 13.06 per cent oxygen, and 49.15 per cent nitrogen. Another sample from this place carried 42.77 per cent methane, 1.51 per cent carbon dioxide, and 11.11 per cent oxygen. The Baltimore return was a split from the south workings in that bed where much coal was being mined in an area a mile long by 1,200 feet wide from north to south. The bed here is 16 feet thick. The Hillman return air represents only a part of the ventilation from that bed as some of it goes out by another exit. The return air from the Abbott and Kidney beds, which was probably sampled on the crest of the main anticline, was from a large area of old workings, part of which was closed by falls and from extensive fresh workings on the anticline.

MINES NEAR NANTICOKE AND HANOVER.

AUCHINCLOSS MINE.

The relatively new Auchincloss mine of the Delaware, Lackawanna & Western Coal Co. is in the southeastern margin of Nanticoke near the center of the main basin. The strata are steeply folded and on the south side of the mine are traversed by a great "fault," or abrupt overturn, by which the lower beds are carried up several hundred feet, as shown in figure 31.

All the usual beds were being worked in this mine at the time of the author's visit, except the Red Ash which, however, had been opened to a small extent at the foot of a shaft 1,750 feet deep. The mine is gaseous, all the beds producing considerable methane, but seemingly the volume of gas had no relation to the structure, except that the Baltimore bed was especially gaseous at the steep upturn near the tunnel shown in the middle of figure 31. Feeders occur in places, but most of the gas emanation was from the working faces. One feeder observed was in the Mills bed near the bottom of a small syncline. The Baltimore workings were the most gaseous. The George bed in a place where the strata are nearly vertical reaches the surface in a railroad cut a short distance southwest of the shaft, and gas which has been flowing out of it for years would burn over an area of 10

square feet. There was said to be less than 1 per cent of methane in the upcast air from this mine which, however, carried about 200,000 cubic feet of air a minute.

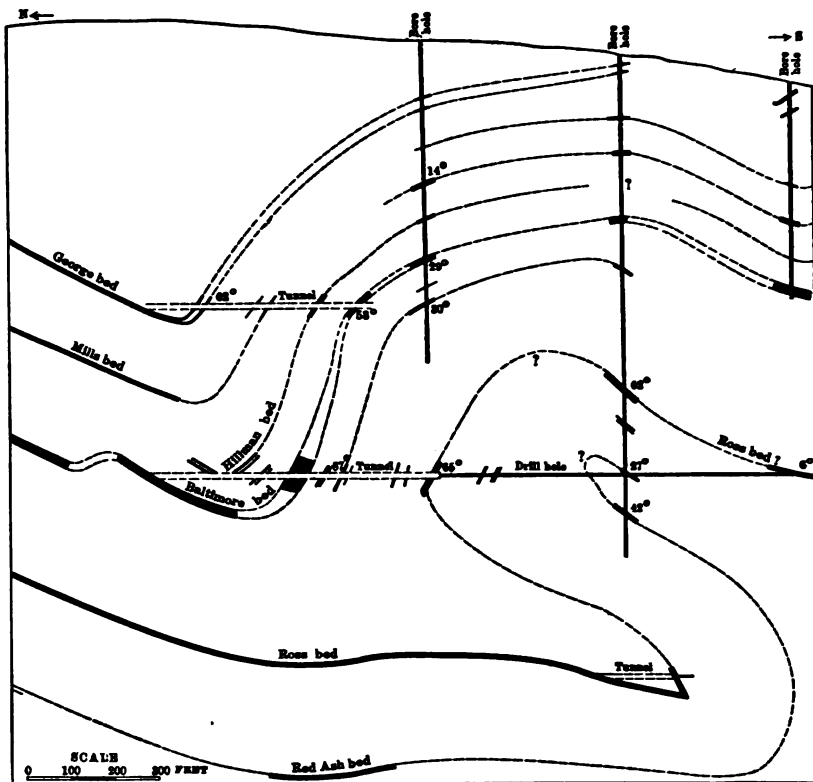


FIGURE 31.—Section through Auchincloss mine near Nanticoke, Pa., showing the great overturn in the lower beds.

Some results of analyses of return air in the Auchincloss mine, made in 1909 by H. J. Force, chemist of the company, are reported as follows:

Results of analyses of return-air samples collected October and November, 1909, in Auchincloss mine.

Shaft.	Return.	Date.	Proportion of methane in sample.	Date.	Proportion of methane in sample.
No. 1.....	No. 3 slope.....	1909 Oct. 8	Per cent. 0.4	1909 Nov. 30	Per cent. 1.0
Do.....	Hillman.....	do.	1.1	do.	.1
Do.....	Upcast at shaft.....	do.	1.2	do.	.8
Do.....	Baltimore slope.....	do.	1.0	do.	.6
No. 2.....	West Mills.....	Nov. 26	2.1	do.	.7
Do.....	East Mills.....	do.	.4	do.	.1
Do.....	Baltimore.....	do.	.7	do.	.8
Do.....	Ross slope.....	do.	.4	do.	.6
Do.....	Ross No. 2 west.....	do.	.2	do.	.7
Do.....	Forge.....	do.	.8	do.	.1

Several samples taken in March, 1912, by H. I. Smith were analyzed in the Pittsburgh laboratories of the Bureau of Mines with the following results:

Results of analyses of return-air samples collected in March, 1912, from the Auchincloss mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in return air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
2503	Ross.....	150 feet west of No. 2 shaft	<i>Cu. ft.</i> 58,300	<i>Feet.</i> 798	<i>Per ct.</i> 0.42	<i>Per ct.</i> 0.07	<i>Per ct.</i> 20.61	<i>Cu. ft.</i> 245
2504	do.....	do. ^a			.42	.08	20.41	245
2491	Baltimore-Mills...	No. 23 tunnel, Mills to George.	44,500	707	2.50	.05	20.37	1,112
2494	Mills.....	On air bridge over east Mills gangway, 150 feet from No. 2 shaft.	37,800	485	.67	.06	20.64	253

^a Duplicate of preceding sample.

Some of the workings in the Auchincloss mine have been very gaseous, but the emissions of gas have not been confined to any one bed. Samples 2503 and 2504 represent the air from the Ross workings tributary to No. 2 shaft, an area nearly 5,000 feet long east and west, and 2,000 feet wide north and south, and in part 900 feet below sea level. Sample 2491 was taken from a return from a relatively small area of the Mills and Baltimore workings on the steep upturn or "fault" south of the shaft. The Mills workings consisted of 17 chambers, a gangway, and an airway; the bed is about 8 feet thick. The Baltimore workings consisted of four chambers, a gangway, and an airway, and were exceedingly gaseous, there being several large blowers in the small area mentioned. The bed is 23 feet thick and the thickness of cover is about 720 feet.

It is believed that the large volume of gas from the small area was due to the crushed condition of the beds in the steep upturn and to their isolation which prevented their being drained of gas to any great degree.

The Mills return was from workings 3,000 feet long with one tier of chambers along steeply dipping beds on the south dip of a syncline. These workings lie deeper than those represented by samples 2503 and 2504. For a long time at the Auchincloss mine a comparison was made between the barometer readings and the amount of gas in the working faces as reported by the fire bosses, but no connection was found.

BLISS MINE.

The Bliss mine of the Delaware, Lackawanna & Western Coal Co. is on the mountain slope south of Nanticoke. The workings extend not only up the slope toward the outcrop but far into the deep basin to the north. There was considerable gas in parts of the Bliss mine,

especially in the lower workings, where all the beds from the Mills to the Red Ash yielded much gas. Little gas was perceptible in the workings on the mountain slope. Several small feeders were observed in the Ross workings in the No. 5 slope.

The results of some analyses of samples collected in various returns in the Bliss mine, made in October and November, 1909, by H. J. Force, chemist of the company, are reported as follows:

Results of analyses of return-air samples collected in October and November, 1909, from Bliss mine.

Bed.	Return.	Proportion of methane in sample.
		<i>Per cent.</i>
Ross.....	Espy tunnel, No. 2 slope.....	0.0
Red Ash.....	Espy tunnel, No. 1 slope.....	.4
Ross.....	Espy tunnel, west Ross off No. 14 tunnel.....	.2
Do.....	Espy tunnel, east Ross off No. 14 tunnel.....	.0
Mills.....	No. 13 slope.....	.2
Do.....	Gangway I.....	.1
Baltimore.....	No. 2 slope.....	.0
Do.....	No. 2 plane.....	.0
Do.....	No. 4 plane.....	.1
Do.....	No. 2 plane.....	.2
Ross.....	No. 1 tunnel.....	.2
Twin.....	No. 2 plane.....	.1
Ross.....	No. 3 plane.....	Trace.
Do.....	No. 9 plane and No. 7 tunnel.....	.9
Ross-Baltimore.....	No. 9 slope.....	.4
Red Ash.....	New plane off gangway F.....	.3

A series of samples collected by H. I. Smith on March 26, 1912, were analyzed in the Pittsburgh laboratory of the Bureau of Mines with the following results:

Results of analyses of return-air samples collected in March, 1912, in Bliss mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in return air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
2496	Ross-Red Ash....	West return 100 feet west of main return.	<i>Cu. ft.</i> 86,100	<i>Feet.</i> 926	<i>Per ct.</i> 0.03	<i>Per ct.</i> 0.02	<i>Per ct.</i> 20.82	<i>Cu. ft.</i> 26
2497	Ross.....	East return 100 feet east of Bliss shaft.	49,280	440	.89	.07	20.60	438
2498	do.....	do. a.....			.92	.06	20.49	453
2499	Baltimore-Forge-Twin.	No. 3 slope.....	46,600	1,080	.76	.05	20.51	354
2501	do.....	Gangway F on bridge....	68,600	980	0	.05	20.90	0
2502	Mills-Hillman.....	On Mills return 200 feet from shaft.	36,600	530	0	.06	20.86	0

a Duplicate of preceding sample.

The analysis of sample 2496 shows that the Ross and Red Ash workings in the west half of the Bliss mine gave off little methane. These workings extend over about 90 acres and although the Ross has been worked out the Red Ash is being actively mined. The Ross return air, represented by samples 2497 and 2498, was from about

400 acres of old workings east of the shaft which were exceedingly gaseous when the coal was being mined, and at the time the samples were taken were giving off a fair volume of methane. The No. 3 slope return current on Baltimore bed was a split ventilating workings 1½ miles long by one-half mile wide in the Baltimore, Forge, and Twin beds. The Baltimore return air, sampled on the bridge on gangway F, was from a large area, about 200 acres, that lies in the south basin. The measures rise on either side and also diminished in depth to the west from a depth of 430 feet at the shaft. The bed is not gaseous, and as it lies in a high basin, much of its gas must have drained off long ago. The Mills-Hillman return was from an area three-fourths of a mile long by one-half mile wide, extending from the outcrop on the south to the fault on the north, and having only 150 feet or less of cover. The Mills workings are nearly all old and the Hillman workings were being mined by about 15 men at the time sample 2502 was taken.

TRUESDALE MINE.

The Truesdale mine, operated by the Delaware, Lackawanna & Western Coal Co., is in the south slope of the basin northeast of the Bliss mine. A large part of the workings are along the mountain slope where the coal measures are not far beneath the surface, but there are also extensive workings in the deeper basin to the north. No examination was made of the conditions in this mine, but several analyses of the return air are available, one series being taken in 1909 by H. J. Force, chemist of the company, and another March 27, 1912, by H. I. Smith, of the Bureau of Mines. Unfortunately, only part of them were taken in places where the results are comparable. The results were as follows:

Results of analyses of methane in return air in the Truesdale mine, 1909.

[H. J. Force, analyst.]

Date.	Bed.	Return.	Proportion of methane in sample.
			<i>Per cent.</i>
Aug. 21.....	Ross.....	No. 2 shaft, east.....	0.1
Do.....	do.....	No. 2 shaft, No. 1 slope.....	.0
Do.....	do.....	No. 2 shaft, west return.....	.3
Do.....	Red Ash.....	No. 2 shaft, top split.....	.4
Do.....	Forge.....	No. 2 shaft, west return.....	.2
Aug. 20.....	Ross.....	No. 3 slope, No. 1, west gangway.....	1.2
Do.....	do.....	No. 3 slope, bottom.....	.8
Do.....	do.....	No. 3 slope, No. 2, east gangway.....	.8
Do.....	do.....	No. 3 slope, No. 1, east gangway.....	.3
Aug. 19.....	Hillman.....	No. 6 slope.....	.0
Do.....	do.....	Holland Tunnel.....	.0
Do.....	Forge.....	No. 300 gangway.....	.3
Do.....	Mills.....	No. 5 slope.....	.0
Aug. 18.....	do.....	No. 3 east split off No. 1 slope.....	.1
Do.....	do.....	No. 3 east airway off No. 1 slope.....	.1
Do.....	do.....	No. 3 west, No. 1 slope.....	.0
Do.....	do.....	do.....	.2
Do.....	do.....	No. 4 west, No. 1 slope.....	.1

Results of analyses of return-air samples collected in March, 1912, in Truesdale mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in return air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
2548	Red Ash, top and middle.	Topsplit, gangway, No. 1 Tunnel to mountain fan.	<i>Cu. ft.</i> 34,900	<i>Feet.</i> 513	<i>Per ct.</i> 0.00	<i>Per ct.</i> 0.05	<i>Per ct.</i> 20.64	<i>Cu. ft.</i> 0
2549	do.	do. ^a			.01	.08	20.31	3
2552	Ross, top Red Ash.	Main return to No. 1 shaft on Ross.	214,500	1,153	.95	.05	20.01	2,037
2553	do.	do. ^a			.35	.04	20.82	751
2550	Ross, top and bottom.	Under Ross gangway, No. 1 Tunnel	39,000	502	.01	.05	20.72	4
2554	Ross - Red Ash - Twin.	Ross No. 2 return, 150 feet from shaft.	122,900	945	.36	.07	20.79	443
2555	do.	do. ^a			.34	.08	20.56	418
2556	Baltimore Forge.	On Forge, 120 feet from shaft.	66,800	903	.23	.04	20.67	154
2557	do.	do. ^a			.23	.08	20.78	154
2558	Hillman Forge.	No. 6 slope airway between No. 1 east and No. 2 east air shafts.	124,300	1,462	.56	.05	20.54	696
2559	do.	do. ^a			.62	.04	20.78	771
2546	George Mills.	Near No. 1 slope fan (in doorway).	135,500	1,316	.50	.05	20.66	675
2547	do.	do. ^a			.50	.04	20.75	675

^a Duplicate of preceding sample.

The Red Ash workings on the mountain slope are almost free from gas, owing to its escape through the thin cover. The area is large, however, extending 3,500 feet along the slope and nearly one-half a mile north from the outcrop, much of the coal being worked out. The returns from the deeper workings carry much methane, especially the large main Ross return to No. 1 shaft. This return ventilates the No. 3 slope section, which extends more than a mile down the steep dip and three-fourths of a mile laterally, comprising the Ross workings—partly old, partly new—and small new workings in the top Red Ash bed. Two samples taken at the same time, however, varied greatly in their methane content, one showing that the return carried 2,037 cubic feet a minute, the other 751 cubic feet.

Samples 2554 and 2555 were taken from a split of Ross air which ventilated workings in the Ross, Red Ash, and Twin beds in a steep dip west of the No. 2 shaft. Sample 2550 was taken from the same return but in air from Ross workings only, the small methane content of which appears to indicate that the large volume of methane shown by samples 2554 and 2555 came almost wholly from the Red Ash or Twin beds.

The return from the Forge and Baltimore workings ventilated an area of about 200 acres in each bed, all near the outcrop. The Hillman and Forge return represented by samples 2758 and 2759 ventilated a narrow area 2,000 feet long near the outcrop, and the increased

amount of methane in it was due to the gaseous character of the Hillman coal. The George and Mills return also carries much gas. The area, about 200 acres, drained by air represented by sample 2546, is on the Truesdale slope; there are many synclines and anticlines. Mining in the Mills bed was confined to robbing pillars, but the George workings were all in fresh coal and were very gaseous in places.

MINES BETWEEN LUZERNE AND AVONDALE.

Coal-mine workings extend almost continuously along the north dip of the coal measures from Luzerne through Kingston to and beyond Avondale. Only a few of these mines were investigated, however, and the only one tested in detail was the Lance at Plymouth, as described on previous pages. In the middle of March, 1912, H. I. Smith collected samples of some of the returns in the Pettebone, Kingston, Plymouth No. 2, Nottingham, Dodson, and Avondale mines.

HARRY E. MINE.

The workings of the Harry E. mine extend from the mountain slope north of Luzerne southeast to Wyoming Avenue, and are all on the north slope of the main basin.

Most of the especially gaseous parts of the Harry E. mine were in the lower workings toward Wyoming Avenue, where the dip has carried the beds to a considerable depth. Here the Ross and Red Ash beds yielded much gas, both in feeders and in a general emanation, especially from fresh working faces. The Pittston bed which was not extensively worked is less gaseous than the others. The Eleven-foot workings, which had been extended down the dip to within a half mile of Wyoming Avenue were not very gaseous. The Red Ash bed, which averages about 8 feet thick in this mine, is overlain by porous sandstone that carries considerable gas, possibly derived from the coal. In a tunnel it was found that this sandstone was so gaseous that the methane could be lighted in the drill holes. A sample of the fresh rock, broken and sealed in the mine, was sent to R. T. Chamberlin, of the United States Geological Survey, who found that the methane content was only 0.013 of 1 per cent. Doubtless a considerable proportion of the gas content escaped from the rock before it was sealed. The gas extracted by keeping the sample 21 days in a vacuum had the following composition:

Composition of gas in sandstone rock.

	Per cent.
Methane.....	4.2
Carbon dioxide.....	1.9
Air.....	70.0
Nitrogen (excess over air).....	23.7

PETTEBONE MINE.

The Pettebone mine of the Delaware, Lackawanna & Western Coal Co. is in the center of the main basin of the coal field, where the general synclinal structure is complicated by a sharp parallel anticline or "fault." All the coal beds are present, the Red Ash being about 1,150 feet below the surface near the center of the basin. There are numerous small crumples in the mine, and some of these have been especially gaseous. In mining the most methane was found in the Hillman bed, as at the Dorrance and other mines in the district. The Red Ash and Cooper beds contain considerable methane, but the Ross yielded very little. The greatest amounts of gas are given off in the deeper part of the basin, near the river.

The author found much methane in many Hillman headings, not only in disturbed or flexed areas, but also in places where the coal was lying practically level. One series of strong "blowers" was found in a syncline adjacent to a steep upturn in which the beds rise 78 feet, the gas coming through the rock floor from the coal below.

A special series of tests for methane in some of the returns was made by the company on August 17, 1909, the results of which have been reported as follows:

Results of analyses of methane in return air, Pettebone mine, August 17, 1909.

Return.	Shaft.	Proportion of methane in return air.
		<i>Per cent.</i>
Red Ash.....	No. 2 shaft.....	1.0
Cooper, south side.....	No. 1 shaft.....	.6
Baltimore.....	do.....	.2
Five-foot.....	do.....	.6
Hillman.....	do.....	1.5
Skidmore.....	No. 2 shaft.....	.1

A few special samples of return air were taken by H. I. Smith in the Pettebone mine on March 14, 1912, and analyzed in laboratories of the Bureau of Mines at Pittsburgh, with the following results:

Analyses of return-air samples collected March, 1912, from Pettebone mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in mine air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
2388	Red Ash.....	East return, 300 feet from shaft.	<i>Cu. ft.</i> 27,400	<i>Feet.</i> 669	<i>Per ct.</i> 0.36	<i>Per ct.</i> 0.05		<i>Cu. ft.</i> 99
2392	Bennett-Cooper...	25 feet east of No. 1 shaft.	39,600	900	.48	.06		190
2393	do.....	do.....			.51	.04	19.78	202
2390	Five-foot-Cooper..	Air bridge, 100 feet from shaft.	29,900	398	.38	.08		114
2391	do.....	do.....			.45	.05		134
2396	Five-foot.....	East of shaft No. 1.....	37,900	903	.97	.05		368
2387	do.....	do.....			1.01	.05	20.55	383
2384	Hillman-Kidney...	On Hillman, 50 feet west of shaft No. 1.	59,700	905	.35	.05	20.76	209
2385	do.....	do.....			.73	.05	20.50	440

* Duplicate of preceding sample.

These samples represented only parts of the ventilation of the mine. The east Red Ash return was from old workings north of the shaft in an area nearly a mile long and lying over 1,100 feet deep in places. The Bennett-Cooper return current was a split from about 70 acres of old workings north of the shaft. The Five-foot-Cooper return ventilated a large area of workings, mostly old, and about 20 fresh places in the Five-foot bed. The intake air was a split which first ventilated the Five-foot workings and then the Cooper. The Five-foot return air was a single split ventilating about 40 acres of live workings north of the shaft and carried increased amounts of gas because fresh coal was being extensively mined. The Hillman-Kidney return was from live workings south of the shaft in an area of 40 acres and the increased amount of methane was probably due to the amount of fresh coal exposed.

KINGSTON NO. 4 MINE.

The Kingston No. 4 mine is one of the large double-shaft mines of the Kingston Coal Co., situated on the slopes west of Kingston. At the time of the author's visit the workings had been extended far up the mountain slope to the north and under the valley flats to the south. All the beds were being worked, some more extensively than others, and there were wide areas of old workings. Methane is given off at many places, but the amount varies greatly from time to time and from place to place in the same bed. In general the strata have a uniform dip to the southeast, and although structural irregularities occur at intervals, they are not large, and the workings at these places have not been especially gaseous. The author did not examine the conditions in this mine, but on March 7 and 9, 1912, H. I. Smith collected samples in some of the returns, which were analyzed with the following results:

Results of analyses of return-air samples collected in March, 1912, from Kingston No. 4 mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in return air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
2344	Red Ash.....	20 feet from No. 4 air shaft.	<i>Cu. ft.</i> 45,500	<i>Feet.</i> 510	<i>Per ct.</i> 0.36	<i>Per ct.</i> 0.12	<i>Per ct.</i> 20.06	<i>Cu. ft.</i> 164
2345	do.....	do. ^a			.30	.05	20.09	137
2360	Cooper.....	Near No. 4 air shaft.....	24,500	490	.04	.14	19.86	10
2361	do.....	do. ^a			.06	.16	19.99	15
2354	Lance.....	Near upper bridge on No. 6 slope.	47,000	627	.13	.08	20.54	51
2353	do.....	At lower bridge over No. 6 slope.	67,100	1,100	.20	.07	20.25	134
2357	Orchard.....	150 feet west of No. 3 shaft.	56,500	1,344	.01		20.54	6
2358	do.....	10 feet south of No. 6 air shaft.	83,500	1,325	.40	.06	20.27	334
2359	do.....	do. ^a			.38	.06	20.33	317

^a Duplicate of preceding sample.

These samples represented only parts of the returns in certain beds and detached sections of the mine, but they afford some interesting comparisons. The Red Ash return air was one split from a large area north of the shaft in which the beds rise on the mountain slope. The workings were all old and largely closed by falls, but many parts were filled with culm flushing, which may account for the large emanation of methane. The sample of air from the Cooper returns was from workings of considerable extent, mostly old, but seemingly yielding little methane. The air in the Lance returns was represented by two parts of one of the main splits ventilating about 100 acres of workings that extended south into the basin. Part of these workings were old, but the deepest ones were fresh and gave off considerable gas. The Orchard return current, which was sampled 150 feet west of No. 3 shaft, was a split draining about 200 acres of workings, some of which were live and some old and mostly caved. The coal is $4\frac{1}{2}$ feet thick, and as the bed is not far below the surface in this district it has lost most of its gas. The sample taken near No. 6 air shaft was from a split from about 100 acres of workings which extend far down into the basin south of the shaft. The upper part of the workings was old, the lower part was fresh workings which lie far beneath the surface, and was very gaseous.

PLYMOUTH NO. 2 MINE.

The Plymouth No. 2 mine, which belongs to the Delaware & Hudson Coal Co., is a short distance east of the Lance mine, but its workings lie higher on the north slope of the basin than most of the Lance workings. The strata dip at low angles to the southeast and rise to the outcrop on the mountain slope to the north. Considerable gas is found in places, especially in the Red Ash bed. The roof is gaseous in places and whenever it cracks considerable gas is given off. A series of samples were taken in the returns of this mine by H. I. Smith on March 22 and 23, 1912, and were analyzed with the following results.

Results of analyses of return-air samples collected in March, 1912, from Plymouth No. 2 mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in return air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
			<i>Cu. ft.</i>	<i>Feet.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Cu. ft.</i>
2513	Bottom Red Ash..	No. 4 gangway, No. 6 plane, 700 feet from No. 5 shaft.	29,300	610	0.17	0.08	20.81	50
2507	Bottom and top Red Ash.	100 feet from No. 2 shaft.	103,600	1,114	.22	.03	20.69	280
2511	Cooper-Five-foot..	Gangway, 150 feet from No. 2 shaft on Cooper.	36,300	595	.10	.04	36.30	36
2515	Stanton.....	400 feet from No. 2 shaft.	45,400	921	.07	.16	20.26	32
2517	do.....	First west gangway on No. 6 slope, 450 feet from No. 2 shaft.	20,600	267	.00	.04	20.91	0
2519	Kidney.....	20 feet from fan.....	26,700	445	.02	.03	20.58	5
2520	do.....	do.....			.01	.03	20.68	3
2509	Snake Island-Abbott.	On Abbott 75 feet from No. 1 shaft.	27,300	290	.07	.03	20.82	19
2510	do.....	do.....			.07	.03	20.76	19

^a Duplicate of preceding sample.

Sample 2507, taken 100 feet from No. 2 shaft, represents the return from a large part of the workings in the Red Ash bed. The methane emanation was of moderate volume for many of the workings were old and some were only moderately deep as the beds rise rapidly to the north up the mountain slope. The Cooper Five-foot return air was a split from the deeper workings in the Cooper bed, about 70 acres in extent, and about 45 acres of up-slope workings in the Five-foot bed. The Stanton return air was in two splits; the first from an area that sloped upward from the shaft and the second, containing no methane, was from deeper workings of moderate extent and about 300 feet below the surface. The Kidney return was from about 20 acres of live workings which were 120 feet below ground, and the Snake Island-Abbott return was from workings similar in extent but somewhat deeper.

Some face samples taken in the return split from the top Red Ash bed contained 52 to 16.60 per cent of methane, the sample containing 16.60 per cent being taken in the No. 2 west gangway off the No. 9 slope.

DODSON MINE.

The Dodson mine in Plymouth is on the north slope of the basin, the workings extending down nearly to its center. The author made no investigation of the conditions in this mine but according to report the mine is less gaseous than the neighboring Lance and Nottingham mines. On March 5, 1912, H. I. Smith collected samples

of return air from three returns in this mine which were analyzed with the following results:

Analyses of return-air samples collected in March, 1912, from the Dodson mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in return air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
			<i>Cu. ft.</i>	<i>Feet.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Cu. ft.</i>
2340	Red Ash-Ross.....	Tunnel from Five-foot to Bennett workings.	50,400	600	0.09	0.05	20.90	45
2343	Hillman-Orchard.....	Entry, regular Hillman return.	62,100	1,080	.05	.27	19.97	31
2338	do.....	Head rock plane from Orchard to Hillman workings.	38,500	876	.10	.40		38
2339	do.....	do.			.07	.42	18.96	27

* Duplicate of preceding sample.

These returns carry only small volumes of methane notwithstanding the fact that they drain extensive workings where there is much mining; the same beds in a similar position in the Lance mine just east carry much larger amounts. These results show the variability in methane occurrence, which in this area is not due to difference in structure or depth, or character of cover.

NOTTINGHAM MINE.

The Nottingham mine of the Lehigh and Wilkes-Barre Co. is in the western part of Plymouth. It was visited by H. I. Smith on February 23, 1912, and several samples of air were taken, which were analyzed with the following results:

Results of analyses of return-air samples collected in March, 1912, from Nottingham mine.

[G. A. Burrell, analyst.]

Lab. No.	Return.	Place of sampling.	Air current.		Proportion of gases in return air.			Volume of methane per minute.
			Volume per minute.	Velocity per minute.	Methane.	Carbon dioxide.	Oxygen.	
			<i>Cu. ft.</i>	<i>Feet.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Per ct.</i>	<i>Cu. ft.</i>
2181	Red Ash.....	75 feet east of No. 2 fan..	52,400	624	0.41	0.13	19.65	215
2183	do.....	East of No. 1 fan.....	63,300	713	.49	.24	19.74	310
2184	do.....	Right slope, 150 feet from No. 2 air shaft.	159,000	914	.73	.18	20.48	1,161
2185	do.....	do.			.66	.14	20.42	1,049
2176	Ross.....	East and west returns at bottom of No. 2 air shaft.	44,200	502	1.59	.06	20.31	703
2174	do.....	100 feet south of No. 2 air shaft.	57,800	359	1.14	.07	20.28	659
2175	do.....	do.			1.14	.05	20.18	659

* Duplicate of preceding sample.

The Red Ash return-air samples were taken from parts of the extensive Red Ash workings, large areas of which are very old. Probably the total emanation of methane from the Red Ash bed in this mine is more than 2,500 cubic feet a minute. The Ross returns represent a large area of deep workings, some of which were in active operation at the time of sampling.

AVONDALE MINE.

The Avondale mine of the Delaware, Lackawanna & Western Coal Co. is on the north bank of the Susquehanna about halfway between Plymouth and Nanticoke. The workings extend far to the south under the river flats and also under the mountain slopes to the north. In the deepest workings the Red Ash bed is more than 400 feet below sea level, or about 900 feet below the river. The lower workings have always been gaseous and considerable gas has been found in places on the higher slopes.

A few years ago part of the deeper workings suffered a serious squeeze, and then, partly on account of the fracturing and partly on account of ventilation being impeded, a large amount of gas accumulated in the lower levels. A series of samples of return air from workings in the affected territory was taken by the company July 24, 1909, and analyzed by H. J. Force, chemist of the company, with the following results:

Methane in samples of return air from workings affected by squeeze.

	Per cent.
Red Ash bed, No. 6 slope.....	1.0
Ross bed, No. 5 slope, west side.....	1.6
Ross bed, No. 5 slope, east side.....	1.2

On March 15, 1912, the mine was visited by H. I. Smith, who collected two samples of part of the return air from Red Ash workings under the flats, the workings being mostly old and about 100 acres in extent. The return on No. 2 slope in the Red Ash bed, 600 feet from the fan, was passing 117,600 cubic feet of air a minute, which carried 0.79 per cent methane, or 940 cubic feet a minute. This sample contained 0.10 per cent of carbon dioxide. A sample taken in No. 7 air slope, 300 feet from No. 1 Ross fan, where the air volume was 29,200 cubic feet a minute, contained 0.17 per cent of methane and 0.07 per cent carbon dioxide.

METHANE FROM CULM.

It is a remarkable fact that culm flushed into the mine workings gives off methane, sometimes in sufficient volume to be dangerous. The gas escapes not only while the culm is wet, but also after the water has drained off. Much of the culm that is used for filling has been lying in great heaps for 20 years or more. Culm consists of coal

mixed with more or less "slate" and lately is being utilized by crushing and sorting out the coal of saleable size. The refuse from this process, consisting of a mixture of fine coal, "slate," and dirt, is flushed into the old mine chambers, so that the pillars can be removed. At most mines the fresh waste from the breakers is also treated by the same process, so that it is not weathered like the material on the old culm heaps.

No investigation was made as to the relative amounts of gas in the old culm and the new waste as now flushed into the mines, but according to reports all the material gives off some methane. This fact is in accord with the chemical investigations of coal samples described in another part of this report for these were found to give off gas for several years after they had been collected.

PRESSURE OF GAS IN THE ANTHRACITE FIELD.

It is evident that the gas is under great pressure in the coal in parts of the anthracite field, but no special tests were made to determine the pressure. Some of the gas blowers and feeders show much force, and in many places the coal is thrown from the face in such a manner as to indicate high pressure.

The only pressure observation made was at a hole bored 45 feet from the upper Ross workings into the top of the lower Ross bed in the Maxwell mine at Ashley, which had tapped gas in large volume. This gas was held by a valve placed in a casing which had been cemented tightly in the rock. The author had a steam gage attached to the valve, and the next day found that it registered 45 pounds to the square inch.

GAS IN BORE HOLES IN THE ANTHRACITE FIELD.

There is no more direct proof of the variation in the amount of gas in the coal or at least of the rate of emanation than the varying amounts of gas given off by bore holes. Thousands of holes have been sunk in the anthracite area, and although most of them do not give off much gas, others have tapped strong flows. Occurrences of this character are not related to depth or structure of the coal, although in some cases the gas comes from a bed or district known to be gaseous. Some of these holes are sunk in the mines from workings in one bed to other coal beds.

One noteworthy instance of this character is in the Maxwell mine at Ashley where, as described above, a hole from the upper Ross workings to the top of the lower Ross bed struck a large volume of gas which had a pressure of 45 pounds to the square inch. In many cases the gas flow is short lived, but in others it continues for a long time. In the Lance mine a 10-inch hole from the Stanton workings to

the Red Ash bed developed a flow of gas estimated at 1,000 cubic feet a minute, and blower gas has been used for fuel in this mine.

In the Holden mine at Taylor a bore hole from the Clark bed through the Dunmore beds yielded a large flow of gas, which at one time was piped to the surface and used for fuel and in the Stanton mine near Wilkes-Barre a feeder from the Hillman (?) bed was piped to a pump house and used for illumination for many years. A bore hole from the Clark workings to the Dunmore bed in the Pancoast mine near Scranton also yielded a large volume of gas. Gas was found in borings from the surface through the coal measures at Hudson, and has been flowing out of the casing for several years; when it is ignited the high flame indicates a large volume.

Some of the bore holes through the valley filling into the coal measures near Wilkes-Barre and elsewhere have tapped gas in considerable volume. One such hole is a few rods southeast of Luzerne Station, where a pipe from a 98-foot hole sunk 11 feet into shale has been giving off gas for years. A sample of this gas was collected and analyzed by Chamberlin ^a with the following results:

Results of analysis of gas from bore hole near Luzerne Station.

	Per cent.
Methane.....	91.31
Olefins.....	.49
Carbon dioxide.....	1.59
Carbon monoxide.....	.04
Hydrogen.....	.82
Air.....	1.10
Nitrogen (excess).....	4.65
	100.00

A sample of gas from a bore hole at the foot of the No. 11 slope in the Dorrance mine was also collected on September 4, 1908, and analyzed by Chamberlin with the following results:

Results of analysis of gas from bore hole in Dorrance mine.

	Per cent.
Methane.....	90.65
Olefins.....	.09
Carbon dioxide.....	.05
Carbon monoxide.....	.00
Hydrogen.....	.77
Air.....	2.73
Nitrogen (excess).....	5.71
	100.00

^a Chamberlin, R. T., Notes on explosive mine gases and dusts, with special reference to explosions in the Monongah, Darr, and Naomical coal mines: Bull. 26, Bureau of Mines, p. 31; Reprint of U. S. Geol. Survey Bull. 383.

GAS CONDITIONS IN ANTHRACITE FIELD COMPARED WITH THOSE IN OTHER AMERICAN MINES.

Little information is available as to the amount of methane given off in mining coal in the United States, but in the course of investigations by the members of the Bureau of Mines return-air samples have been obtained at several mines. Analyses of some of these presented in the following tables are of interest for comparison with the analyses given on the preceding pages. All the analyses were made by G. A. Burrell, chemist of the bureau.

Analyses of part of return air from mine at Roslyn, Wash.

[Samples collected by H. M. Wolfelin.]

Lab. No.	Return.	Volume a minute.	Velocity a minute.	Methane.		Carbon dioxide.	Oxygen.	Nitrogen.
				Per cent.	Cubic feet a minute.			
		<i>Cu. ft.</i>	<i>Feet.</i>			<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
1431	Main.....	39,260	755	0.39	153	0.07	20.48	79.06
1434	Seventh east split.....	4,450	100	1.28	57	.07	20.36	78.29
1436	Eighth west.....	1,592	70	1.33	21	.06	20.25	78.36

This mine is considered gaseous, but the methane volume in the main return would be regarded as moderate in the anthracite region.

Methane in various parts of Ellsworth 3 and 4 mines in Pittsburgh coal bed, Washington County, Pa.

[Samples collected by Dunbar, Mar. 3, 1911.]

Place where sample was taken.	Methane.	
	Cubic feet a minute.	Per cent.
Entries at shaft.....	160	0.19
Main return.....	208	.70
Left entry, working face ahead of air.....		1.38
Left west main entry, 30 feet beyond air.....		1.80
South split face.....	228	.96
West split face.....	38	.17
West main split.....	152	.68
Split D face.....	52	.39
Northeast split, C face.....	207	.94
West main split.....	165	1.40
North split, C face.....	35	.32
Northeast split, C face.....	261	.75
East main parallel entry.....	17	.10
9th west split, off A face.....	22	.14
C face, south split.....	308	.96
Back of shaft 3.....	223	.71
D face, west split.....	27	.34
C face, northeast split.....	22	.10
Main return, west of shaft 4.....	450	.75
Return to Tombaugh air shaft.....	321	.86

These methane determinations show great variation in the emanation in different returns and from a number of working faces. The variations are not related to the amount of coal mined, and indicate

that the same bed of coal is giving off much more gas at one place than another. The bed is thick, seemingly uniform in character, and lies nearly horizontal.

In some mines there is no appreciable trace of methane in the returns. One instance of this is in the Bevier coal field in Macon County, Mo., where two samples of the return air were collected by H. I. Smith, April 1, 1911. One return 200 feet south of the shaft with a volume of 15,732 cubic feet a minute contained no methane and 0.39 per cent of carbon dioxide. Another return with 21,610 feet of air was also free from methane and contained 0.44 per cent of carbon dioxide.

SOME FEATURES OF FIRE-DAMP EXPLOSIONS IN THE NORTHERN ANTHRACITE FIELD.

Explosions of small bodies of fire damp frequently occur in the deeper mines in the anthracite region, many of them with fatal results. According to the reports of the Department of Mines of Pennsylvania 245 men were killed by gas explosions in the northern anthracite basin in the 10 years from 1899 to 1909, an average of nearly 25 men a year. It is not the purpose of this report to take up the cause and history of these accidents, but a few statements will be given relative to the manner in which the fire damp occurs.

The gas explosions take place under a great variety of conditions, but most of them are caused by taking an open light into a place where there is an unexpected body of fire damp. Generally such accumulations are at the working faces but sometimes are in old workings. In many instances, however, the existence of gas is known but warnings are disregarded and by striking a match, opening a safety lamp, or carrying an open light the gas is ignited.

Sometimes when new workings break through into old ones a body of gas is encountered. The temporary lessening of air currents by leaving doors open or breaking down the brattice by blasts is a frequent cause of the accumulation of gas at the face, and then, when circulation is restored, the accumulation of gas is carried out into the workings and perhaps ignited. Gas given off by a run of coal has caused many an explosion, for a large body of broken coal may give off a large volume of gas quickly. There have been some noteworthy outbursts of this class in those sections of the anthracite region where the beds dip steeply. Many blowers and feeders are opened where little gas was given off before, and the silent emission of gas from the face or roof sometimes fills a chamber very quickly. It is a matter of daily observation that although the fire boss may find no gas in the working place in his early morning inspection, by the time the miners arrive gas has accumulated to a dangerous extent, even though the ventilation is carried reasonably close to the face. Gas often

accumulates when the air is still in old workings and generally in roof cavities, where a man climbing over the pile of fallen material with an open light in his cap may ignite it.

A few quotations from records of ordinary explosions are here given to illustrate some of the conditions of occurrence of unexpected bodies of fire damp.

(1) Regarding an explosion in which 7 men were fatally burned and others more or less injured, a district mine inspector stated that he "made a thorough investigation as to the cause of the explosion but could find none. The ventilation is good but it may be that some doors were allowed to remain open on the day of the accident."

The jury examined the place and finding no evidence of cause, stated in the verdict that the mine was properly managed, the ventilators were good and the mine laws strictly complied with in every particular. The gas accumulated very rapidly in certain chambers but the cause of this accumulation is unknown. It was ignited by an open lamp carried by a man who had no expectation of encountering the gas.

(2) * * * miner fatally burned by gas. He turned to the face of his breast after firing a blast which cut a feeder of gas and the gas was ignited by an open light on his head.

(3) * * * miner fatally burned by gas. He fired a blast in the face of the breast which broke the brattice down. He kept on working at the face and gas accumulated, which in a short time was ignited by his open light.

(4) At the — colliery a small explosion of gas in a chamber in the Ross bed killed one man, burned another one somewhat, and caused a fire that suffocated 8 other men. The chamber was driven about 400 feet on a 5 per cent dip to a point where there is an abrupt upturn. The coal here is greatly shattered and apparently gave off a puff of gas causing the explosion. Very little gas was made driving to this point.

(5) The chamber was found free of gas by the fire boss in his morning round. Later a rock fall broke the brattice and cut off air from the face. A miner climbed over the fall with naked light and ignited a small body of gas which had accumulated in the meanwhile. He was fatally burned by the explosion.

(6) — was fatally injured in the Five-foot vein. He was driving a crosscut into an abandoned chamber. He fired a blast and evidently broke through and then went in with naked light to see what the shot had done. When near the face he ignited the gas.

(7) Miner killed by explosion in — mine. He had been working in a chamber about 40 feet above the gangway, and ignited gas that had accumulated while he was trying to set off a blast that would not go off. Meanwhile he left open the door controlling the ventilation.

(8) An explosion in the Cooper bed killed 6 men. The place was reported free from gas in two trips of the fire boss. In the afternoon a miner engaged in driving a line chamber fired a blast in the face which caused an explosion, killing him and 5 other men who were within 300 feet.

(9) At — mine someone set fire to some feeders in an old chamber. An hour later while inspecting the place there was an explosion which killed 2 men. In this same mine later in the day a man returned to his face after an absence of a half hour and in lighting a match ignited gas that burned him fatally. On another occasion in this mine a man ignited gas by opening his safety lamp and was killed by the explosion.

EXPLOSIVE GAS IN MINES IN THE SOUTHERN PART OF THE ILLINOIS COAL FIELD.

INTRODUCTION.

In the winter of 1912 an investigation was made by the Bureau of Mines of some of the conditions under which gas occurs in the principal mines in a part of the Illinois coal field. The work was done under a cooperative agreement with the State geological survey and the department of mining engineering of the University of Illinois.^a The region selected was the part of the field lying south of latitude 38° and extending from Murphysboro on the west to Eldorado on the east. The outcrop zone of the coal was the southern limit. This district is 50 miles long and about 18 miles wide, its area thus being nearly 900 square miles. Nearly all the mines in this area were inspected as regards the amount and the mode of occurrence of gas and in 56 of them samples of the return air current were taken to ascertain the volume of gas given off. In some of the mines the total returns were sampled; in others, the returns from different workings were collected separately; and in a number of mines the sampling was repeated several times so as to check results and determine whether varying conditions caused related variations in the volume of gas. Tests were also made of the pressure of the gas in the solid coal by inserting iron tubes into the face, suitably tamping them, and reading the pressures on a gage. In some mines the volume of gas in the coal was determined at the holes.

GENERAL FEATURES OF THE COAL MEASURES IN SOUTHERN ILLINOIS.

DISTRIBUTION AND RELATION OF THE COAL BEDS.

Coal-bearing strata underlie more than two-thirds of Illinois, and the same field, the central, extends far to the eastward in Indiana. The coal measures are estimated to have a total thickness of 2,200 feet and in places there are several thick beds of coal in the succession. However, a thickness of 700 feet of strata at the bottom of the coal measures is not known to contain any coal over 2 feet thick. The general structure is that of a shallow basin with low dips and very few flexures or fractures of the strata.

Near the middle of the Illinois basin the lower coal beds are in places 2,000 feet or more below the surface. Near latitude 35° they lie 400 to 600 feet deep but the gradual rise to the south brings them

^a For scope and purpose of investigation see Preliminary report on organization and methods of investigations: Illinois Coal Mining Investigations, cooperative agreement, 1913, 71 pp.

to the surface in an outcrop zone passing near Murphysboro, Carbondale, Marion, and Harrisburg. The geology and other features of the district treated in this bulletin have been described by DeWolf,^a by Savage and Shaw,^b and by others. A brief résumé of the principal facts is presented here because it will be serviceable in showing the conditions under which the coals occur.

STRATIGRAPHY.

There are three workable coal beds underlying the region, all in the Carbondale formation, which has a thickness of 250 to 300 feet. The top of this formation is the Herrin bed (No. 6); from 90 to 140 feet below is the Harrisburg coal (No. 5), and at the base is the Murphysboro coal (No. 2).

A typical section of the Carbondale formation as given in the record of a bore hole in the western part of the Herrin quadrangle is as follows:

Record of bore hole in Carbondale formation, Herrin County, Ill.

	Feet.		Feet.
Coal No. 6 (Herrin bed).....	8½	Sandstone.....	25
Underclay.....	1	Shale.....	10
Limestone.....	6	Limestone.....	1
Shale.....	14	"Slate".....	3
Limestone.....	3	Coal.....	½
Slate, black.....	4½	Underclay.....	2½
Coal No. 5 (Harrisburg bed).....	4½	Shale, sandy.....	5
Shale.....	58	Sandstone.....	23
Limestone.....	1	Shale.....	24½
Shale.....	3½	Coal (top Murphysboro bed?).....	1½
Coal.....	2	Shale.....	20½
Underclay.....	10½	Coal (lower Murphysboro bed?).....	2½
Shale, sandy.....	22	Shale and sandstone of Pottsville age.	

Above the Carbondale is the McLeansboro formation which underlies the surface deposits in a wide area extending far north from the outcrop of the No. 6 coal. At its base is a gray shale or shaly sandstone that lies on the No. 6 coal and varies from 15 to 110 feet thick, the average of 120 records being 64 feet. In most of the region, especially in the eastern part of Williamson County, the black shale of the roof over the coal is capped by 3 to 4 feet of limestone. About 105 to 110 feet above the No. 6 coal (Herrin) is a fairly persistent coal bed, 1½ to 2 feet thick, in sandy shale and sandstone, and other thin beds of coal occur higher up. The Herrin coal varies from 6½ to 14 feet thick, the average of 130 records being

^a DeWolf, F. W., Coal investigations in the Saline-Gallatin field, Illinois, and the adjoining area: Illinois Geol. Survey Bull. 8, 1907, pp. 211-230, and U. S. Geol. Survey Bull. 316, 1907, pp. 116-136; Coal investigations in Saline and Williamson Counties, Illinois: Illinois Geol. Survey Bull. 8, 1906, pp. 231-245.

^b Shaw, E. W. and Savage, T. E., Description of the Murphysboro and Herrin quadrangles: U. S. Geol. Survey Folio 185, 1912, 15 pp.

9½ feet; at Galatia the average is 6 feet in most places. Eighteen to 30 inches above the base of the bed there is an impure layer known as "blue band," which is a characteristic feature of this bed. This layer is generally nearly 2 inches thick, but near the western outcrop zone it thickens to 6 to 11 inches.

The Harrisburg or No. 5 coal, which underlies the area from Jackson to Saline County, is 5 to 7 feet thick in the mines in Saline County; the average of 40 measurements in the Herrin district is 4½ feet. It is a solid body of massive coal of moderate hardness with few impurities.

It is overlain by a member of black shale, thin limestone, and sandstone, which separates it from the Herrin coal. The thickness of this intervening series is 37 to 40 feet in the Herrin district and westward, but in Saline County it varies from 100 to 140 feet and the succession of beds also varies considerably.

The Murphysboro or No. 2 coal lies about 250 feet below the Herrin coal with sandstone and shales intervening. The coal varies in thickness from 1 to 6 feet and in the greater part of the district is in two beds, which are separated by a body of shale. In the southwestern part of sec. 32, T. 8 S., R. 2 W., the intervening shale is 35 feet thick, whereas in the southeastern part of the section it is only 4 to 5 feet thick. The coal is capped by 20 to 40 feet of clay and shale, which is overlain by 15 to 45 feet of sandstone that has been designated as the Vergennes member of the Carbondale formation. Coal No. 2 is extensively worked about Murphysboro but it is not mined in the region east where it is not far below the lower workings of many mines. At the Gus Blair mine the No. 2 bed consists of 11 inches of top coal, 3 inches of bone, 23 inches of shale and concretions, and 46 inches of coal on a sandstone floor. In a bore hole at Murphysboro the coal in the bed is 6½ feet thick, but to the northward it thins and is absent in places.

STRUCTURE.

As one purpose of the investigation was to ascertain whether or not occurrences of gas in coal have any relation to structure, special consideration was given to the structural features of the coal beds in southern Illinois. A map (Pl. VII) was compiled from maps by Shaw, Savage, De Wolf, and Cady, to show the configurations of the coal beds, and on it are represented also the depths to coal, the lines of outcrops in part, and figures and circles showing the relative amounts of gas at most of the mines. The structure is shown by 50-foot contour lines which give the position of the coal beds with reference to sea level but without relation to the surface of the land. The three principal coal beds are represented in the separate areas each by contour lines of distinctive character. In general the

structure is the same whichever bed is represented, but the altitude is greater or less according to the interval between the beds. data were available for showing the configuration of No. 2 coal in the Carbondale-Marion region and eastward. The contours given are based on known positions of the coal beds in many mines and bore-hole records, being extended hypothetically across unexplored districts. In the region between Herrin, Christopher, Duquoin and Carbondale the contours are based on 200 records. In some cases identification of beds in the bore-hole records is not entirely certain, and this introduces a possible error at certain localities which can not be corrected from the data available at this time.

In general, the structure in Saline, Williamson, and Franklin Counties is a monocline with low dips to the northward at an average rate of 50 feet to the mile. In Jackson County the strike swings around to the north and from De Soto to Duquoin there is a broad belt in which the dips are to the east. Approaching Duquoin the dips become steeper, and near the center of range 1 W., T. 6 N., the beds rise 350 feet in less than three-fourths of a mile. An anticline of considerable prominence extends along a north-to-south course parallel to this zone of steep dip; it passes through the northeastern corner of Jackson County.

The monocline in Williamson, Franklin, and Saline Counties has many local irregularities in direction and degree of dip and its continuity is interrupted by several low but well-marked anticlines and shallow synclines. These are all shown in Plate VII. True faults or dislocations of the strata are rare, but several small breaks have been found in some of the mines and there is a fault of considerable magnitude northwest of Murphysboro near the boundary line between Jackson and Perry Counties. Near Eldorado and Harrisburg the measures are cut by dikes of igneous rock which have altered the adjoining coal to coke.

CHARACTER OF THE COALS.

The Herrin coal is a high-grade bituminous coal in great favor among fuels in the Middle States markets. The average of 17 analyses by S. W. Parr, of the Illinois State Geological Survey, is given by Shaw and Savage^a as follows:

Averages of analyses of 17 coal samples from Herrin district.

Condition of sample.	Total moisture.	Fixed carbon.	Volatile matter.	Ash.	Sulphur.	Calorific value.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>B. t. u.</i>
As received	9.11	48.79	32.66	9.43	1.65	11,799
Moisture free		53.02	36.18	10.39	1.86	13,004
Moisture and ash free		59.39	40.60			14,507

^a Shaw, E. W., and Savage, T. E., Description of the Murphysboro and Herrin quadrangles: U. S. Geol. Survey Folio 185, 1912, p. 14.

OF PARTS OF

N, WILLIAMSON, AND SALINE COUNTIES

THE SOUTHERN PART
OF THE

LD OF ILLINOIS

FIGURES OF THE OCCURRENCE OF METHANE

By N. H. Darton

BEDS REPRESENTED BY CONTOUR LINES

E. Savage, G. H. Cady and F. W. DeWolf

FOUR INTERVAL 50 FEET

Datum, mean sea level

SCALE:

 Raleigh

LOUISVILLE AND NASHVILLE

C.C. & ST. L. R. R.

BOUNDARY
LINE

The range of composition in these samples was as follows: Moisture, 6.5 to 11 per cent; ash, 6½ to 12 per cent; and sulphur, 0.6 to 3½ per cent; and the heating value varied from 11,079 to 12,144 B. t. u.

An average sample from Galatia on the moisture-free basis contained 37.46 per cent volatile matter, 48.75 per cent fixed carbon, 13.79 per cent ash, and 3.73 per cent sulphur, and had a fuel value of 12,505 B. t. u.^a

Calculated on the ash-free and moisture-free basis, the Murphysboro coal ranges from 53 to 63 per cent fixed carbon, 4 to 7 per cent ash, and 0.6 to 2 per cent sulphur. The average heating value of five samples is given at 14,749 B. t. u. (moisture and ash free), the calorific value of the separate samples ranging from 12,500 to 14,854 B. t. u. The coal ranks high as a fuel and much of it is called "Big Muddy coal."

The Harrisburg coal varies considerably in composition, but most of it is of high grade. The average of a series of samples from O'Garmines Nos. 3, 8, 9, 12, and 14, No. 1 of the Wasson Coal Co., and No. 2 of the Saline County Coal Co. are given by De Wolf^b as follows:

Average of analyses of seven samples^a of coal No. 5 from Saline County, Ill.

[S. W. Parr, analyst.]

	Condition.						Referred to coal "moisture and ash free "		
	As received.			Moisture free.					
	Maximum.	Minimum.	Average.	Maximum.	Minimum.	Average.	Maximum.	Minimum.	Average.
	<i>P. ct.</i>	<i>P. ct.</i>	<i>ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>
Moisture	6.64	4.43	5.90						
Volatile matter	36.20	33.48	34.69	38.52	35.66	36.88			
Fixed carbon	52.82	47.87	50.41	55.25	50.94	53.66			
Ash	10.89	7.17	8.98	11.58	7.62	9.55			
Sulphur	3.30	2.19	2.60	3.52	2.30	2.77			
B. t. u.	12,883	12,159	12,552	13,700	12,942	13,197	14,962	14,830	14,910

^a Samples include entire seam except in 1 mine where 1-inch pyrite ball was excluded.

OCURRENCE OF GAS IN SOUTHERN ILLINOIS MINES.

In many of the Illinois mines the coal contains little explosive gas and it is given off so regularly and in such moderate volume that satisfactory ventilation is easily effected.^c This is particularly the case near the outcrop zone where the cover is thin and much of the gas has drained off into the atmosphere. Many of the deeper mines

^a De Wolf, F. W., Coal investigations in Salem and Williamson Counties, Illinois: Illinois Geol. Survey Bull. 8, 1907, p. 245.

^b De Wolf, F. W., op. cit., p. 228.

^c See Parr, S. W., and Barker, Perry, The occluded gases in coal: University of Illinois, Bull. 32, 1909, 28 pp.

in Franklin, Williamson, Perry, and Saline Counties, however, have considerable gas and in some instances it has exploded with serious results. The gas in this area occurs irregularly in the coal and the variations appear not to be closely related to depth of the coal bed or to structure of the strata. The amount of gas also varies considerably in different parts of the mines and at different stages of progress, being closely similar in these respects to gas occurrence in other coal-mining districts. There is a general relation between methane volume and depth up to 200 feet, with many local exceptions, however, when close comparison is attempted. In sampling the return air in the mines care was taken to select such returns as might be expected to show variations in the volume of gas that could be related to variations in conditions in the workings, but in many cases such relations could not be recognized. A sufficient number were obtained to illustrate very clearly the irregularity of occurrence of the gas even if they do not indicate the cause of the irregularity.

CONDITIONS IN REPRESENTATIVE MINES IN HERRIN BED.

MINES NEAR BENTON.

Two mines of considerable size have been worked for several years in the vicinity of Benton. They are both south of the city and reach the coal at depths slightly more than 600 feet. The beds lie nearly horizontal in broad undulations of the general monocline.

HART-WILLIAMS MINE.

The Hart-Williams mine, 2 miles south of Benton, reaches the coal at a depth of 640 feet, or about 175 feet below sea level. The bed is 10 feet thick. There is considerable methane throughout the workings, but the vigorous ventilation removes it. Occasional feeders are encountered, and gas enough to ignite escapes from all drill holes. The southeast workings are most gaseous, due perhaps to the fact that the beds there are traversed by many small rolls.

The returns of the mine were examined in 1912, on January 15 and 16, February 14, and in part on February 27. The air in the northeast return near the shaft was sampled January 15 and 16. It ventilated an area worked by 60 miners. The volume of air on the first test was 25,687 cubic feet per minute, and on the second test 23,120 feet. The percentage of methane was 0.07 on the first test and 0.06 on the second test, or about 16 cubic feet a minute, and the carbon dioxide content was 0.11 and 0.07 per cent in the two tests. The air of the northwest return had a volume near the shaft of 26,270 cubic feet and contained no methane and 0.04 per cent of carbon dioxide. The volume of air passing in the southwest return near the shaft

was 31,930 cubic feet a minute. The air carried 0.08 per cent of methane, or 25½ cubic feet a minute, and 0.09 per cent of carbon dioxide. The southeast return from workings in which 95 miners were employed had a volume of 26,718 feet of air per minute, containing 0.18 per cent of methane, or 48 cubic feet a minute, and 0.15 per cent of carbon dioxide. The air in the entire east return—north-east and southeast—carried 64 cubic feet of methane a minute, and the air in the upcast from the mine carried 90 cubic feet of methane a minute.

On February 14 the sampling was repeated in part at other places, in order to subdivide the air in greater detail, with the following results:

Methane and carbon dioxide in return air in Hart-Williams mine, February 14, 1912.

Place where sample was taken.	Volume of air per minute.	Proportion of methane in return air.	Volume of methane per minute.	Proportion of carbon dioxide in mine air.
	<i>Cu. ft.</i>	<i>Per cent.</i>	<i>Cu. ft.</i>	<i>Per cent.</i>
First southeast entry, near shaft.....	21,120	0.20	42	0.26
Main east return, near shaft.....	20,830	.11	23	.32
First southwest entry, near shaft.....	15,912	.13	21	.15
East return, off third northwest airway.....	20,016	.17	34	.24
First crosscut, off third northwest airway.....	14,490	.02	3	.06
Main west return at overcast.....	18,480	.19	35	.15
Total upcast at shaft.....	110,848	.12	133	.61

On February 27 a sample taken in the first southeast air course, 350 feet southeast of the shaft, where the volume was 12,600 cubic feet a minute, contained 0.22 per cent methane and 0.17 per cent carbon dioxide.

BENTON MINE.

The Benton mine, on the southwestern margin of Benton, is 630 feet deep at the shaft, and the workings extend across the arch of a low anticline with axis trending nearly east and west. The mine gives off considerable gas, there being more in the western chambers than in the eastern ones.

The principal returns were sampled January 16 and March 9, 1912, while the mine was in full operation, and again in April 15, 1912, 16 days after mining had ceased. The samples were taken from the main north return near the shaft, which included all the north returns from a district in which 90 miners were working, from the main south return near the shaft and from the west return near the shaft, the two being the total return from the south workings. There were 80 miners employed in the area ventilated by the west return and 70 miners in the area ventilated by the main south return.

The results of the analysis of the three samples and their total by addition are as follows:

Methane and carbon dioxide in return air from Benton mine, January, March, and April, 1912.

Return.	Jan. 16, 1912.				Mar. 9, 1912.				Apr. 15, 1912.			
	Volume of air per minute.	Proportion of methane in return air.	Proportion of carbon dioxide in return air.	Volume of methane per minute.	Volume of air per minute.	Proportion of methane in return air.	Proportion of carbon dioxide in return air.	Volume of methane per minute.	Volume of air per minute.	Proportion of methane in return air.	Proportion of carbon dioxide in return air.	Volume of methane per minute.
Total upcast at main shaft.....	Cu. ft. 101,200	P. ct. .20	P. ct. .10	Cu. ft. 96	Cu. ft. 92,290	P. ct. .21	P. ct. .06	Cu. ft. 194	Cu. ft. 57,144	P. ct. .21	P. ct. .06	Cu. ft. 46
Main north.....	47,400	.20	.10	96	38,016	.18	.11	653	19,140	.21	.11	46
West.....	24,400	.18	.10	44	4,840	.34	.13	74	16,800	.49	.20	83
Main south.....	29,400	.18	.13	53	32,434	.16	.16	52	21,204	.20	.15	42

^a Air not circulating in all of north workings owing to repairs; also some leak from overcast.

HANAFORD MINE.

The Hanaford mine at Smothersville is the deepest in the area investigated, as the coal lies 696 feet below the surface at the shaft, or nearly 200 feet below sea level. The mine produces 1,000 tons a day. About 6½ feet of the coal is mined. The return air was measured and sampled February 1, 1912, in the four principal returns.

The air in the first southwest return ventilated workings in which there were 12 miners. The return air, 325 feet west of the shaft, had a volume of 6,000 cubic feet per minute and contained 0.10 per cent of methane, or 6 cubic feet a minute, and 0.09 per cent of carbon dioxide. The main west return, in which a sample was taken at the second south airway, 400 feet west of the shaft, ventilated workings in which 16 miners were employed. It carried 8,520 cubic feet of air per minute, which contained 0.13 per cent methane, or 11 cubic feet a minute, and 0.05 per cent carbon dioxide. The air of the main east return, from workings in which there were 35 miners, had a volume 75 feet east of the shaft of 34,000 cubic feet per minute and contained 0.14 per cent methane, or 48 cubic feet a minute, and 0.05 per cent carbon dioxide. The first northwest return at the overcast 425 feet northwest of the shaft, from workings in which there were 10 miners, carried 7,540 cubic feet of air per minute, containing 0.14 per cent methane, or 10½ cubic feet a minute, and 0.09 per cent carbon dioxide. There was notable uniformity in the percentages of methane in the four returns and close conformity between the volume of methane and the amount of mining on each. However, the volume of methane from the southwest workings was a little less than from those on the northwest and main west returns.

MINES NEAR CHRISTOPHER.

Two mines near Christopher were selected as likely to illustrate the conditions in that part of the field—the mine of the United Mining Co., a mile east of the town, and the Zeigler District mine, a mile north of town.

UNITED MINE.

The United mine is 500 feet deep at the shaft and has a daily production of about 4,300 tons. The return from the north workings was sampled just north of the shaft, and with somewhat uncertain cross section, owing to obstruction of the air passage by cars; it indicated a current of 30,000 cubic feet per minute. This air contained 0.20 per cent of methane and 0.11 of carbon dioxide. The air of the south return, part of which was sampled in an airway just south of the shaft and part at an outlet in a crosscut at the shaft, had a total volume of 80,000 cubic feet a minute, the two samples containing 0.20 and 0.21 per cent of methane and 0.07 and 0.08 of carbon dioxide, respectively. The total volume of methane in the north return was 60 cubic feet a minute and in the south return and crosscut 160 cubic feet, or 220 in all. A part of this mine had a squeeze a few years ago and some of the workings were stopped off. It is known that a large amount of gas has accumulated in the inclosed workings. Samples of this air taken by S. O. Andros, of the department of mining engineering of the University of Illinois, from a 3-inch pipe through a stopping were found to contain 19.75 and 38.17 per cent methane and 0.03 and 0.20 per cent carbon dioxide, respectively.

Andros also obtained some samples of gas from a $\frac{1}{4}$ -inch pipe which was inserted 6 inches into the rib at the sixth northeast entry near the fifth panel. The results of analysis of the two samples were as follows:

Composition of gas from pipe in rib coal in United mine.

Constituent.	Sample 1.	Sample 2.
	<i>Per cent.</i>	<i>Per cent.</i>
Methane.....	24.61	38.74
Carbon dioxide.....	10.10	11.89
Oxygen.....	14.54	11.89
Nitrogen.....	60.75	51.27

Samples of the average air taken by Andros in the sixth northeast cross entry contained 1.08 and 0.59 per cent methane and 0.06 and 0.05 per cent carbon dioxide, respectively. This entry, which has 23 working places on it, was reported to be the most gaseous cross entry in the mine.

ZEIGLER DISTRICT MINE.

The Zeigler District mine, a mile north of Christopher, is 517 feet deep at the shaft and has a production of about 2,000 tons a day.

Considerable gas was reported all through the mine, especially in chambers up the slope, where it has to be cleared out by close bratticing. The coal is 9 feet thick, of which 7 feet 4 inches is taken out. The main south return, in which a sample was taken just south of the main shaft, carried 64,320 cubic feet of air per minute, of which about one-half was from the southeast and one-half from the southwest workings. Part of the return air from the southeast workings escaped by a crosscut into the east side of the shaft, which adds 17,250 feet to the total volume, or 81,570 in all. The proportion of methane in the larger current was 0.19 per cent and in the smaller one 0.16 per cent, and of carbon dioxide 0.09 and 0.10 per cent. The main north return from the northeast and northwest workings, in which samples were taken near the shaft, carried 48,000 cubic feet of air per minute, which contained 0.19 per cent methane and 0.09 per cent carbon dioxide, practically the same proportions as in the south return. The total volumes, however, indicated by the tests were 150 cubic feet of methane a minute in the south return and 91 cubic feet in the north returns, closely comparable with amounts in the United mine.

ZEIGLER MINE.

Owing to explosions and fires several years ago the mine at Zeigler has the reputation of being a gaseous one. It is remote from other mines, the nearest ones being Rend No. 2 and the North mine at Royalton. The coal lies 418 feet below the surface at the shaft and dips at low angles to the north and northeast. The ordinary output of the mine is 2,300 tons a day and 8 or 9 feet of coal are taken out. The workings have an area of about 500 acres and part of them are walled off on account of roof falls caused by the explosions and fires.

The return-air samples were collected on February 2 and March 23 while the mine was under full operation and again on April 7, eight days after mining had ceased. The return air comes to the upcast in two main currents, one from the north side of the mine where there were 75 miners, and the other from the south side where a force of 150 miners was employed under ordinary conditions.

On February 2 the air in the north return near the shaft had a volume of 77,616 cubic feet a minute and contained 0.12 per cent methane and 0.08 per cent carbon dioxide; and in the south return, 84,800 feet, 0.07 per cent methane, and 0.10 per cent carbon dioxide.

On March 23 the north return had a volume of 64,288 cubic feet of air containing 0.24 per cent methane and 0.10 per cent carbon dioxide; the south return, with a volume of 90,396 feet, contained only 0.03 per cent methane and 0.05 per cent carbon dioxide.

Two weeks later, eight days after the mining had ceased and the fan was running less rapidly, the north return was passing 61,992

cubic feet of air, which contained 0.09 per cent methane and 0.05 per cent carbon dioxide. The south return, 700 feet south of the main shaft, was found to carry 57,600 cubic feet of air which contained 0.09 per cent methane, and 0.07 per cent carbon dioxide. Between this point and the shaft there was a leakage of about 15,000 cubic feet of air per minute from a fresh air overcast and a sample taken near the shaft at this time contained 0.03 per cent methane and 0.05 per cent carbon dioxide. This same leakage diluted the south returns in the two previous tests so that the 0.07 per cent of methane and 0.03 per cent of methane are slightly too low to represent the true character of the air. However, if calculation of the number of cubic feet of methane a minute be made from all the figures obtained near the shaft the following results will be obtained:

Volume of methane in returns in Zeigler mine, February to April, 1912.

Return.	Volume per minute.		
	Feb. 2.	Mar. 23.	Apr. 7.
South.....	<i>Cu. ft.</i> 60	<i>Cu. ft.</i> 27	<i>Cu. ft.</i> 22
North.....	93	154	56

In all of the samples of south-side air, it was diluted with about 20 per cent of fresh air. The greatly increased amount of methane in the north return on March 23 could not be accounted for. There was a markedly less amount at all times on the north side, doubtless due to the fact that the number of miners employed on that side was only half as many as on the south side. The only structural difference is that the measures dip to the north and therefore the workings on the north side are slightly deeper than those on the south side.

MINES NEAR DUQUOIN.

The mines east and south of Duquoin produce coal from the Herrin bed. There is an abrupt rise in the strata a short distance east of the city in which the beds ascend nearly 350 feet. At the Majestic mine, 3 miles southeast of Duquoin, the coal is at sea level, or 412 feet below the surface, whereas in the Horn mine, 2 miles southwest of Duquoin, the depth is only 80 feet. In the belief that these diverse conditions of depth would show notable variations in gas in the coal, samples of return air were obtained from the Majestic, Paradise, Davis, and Horn mines. In the Paradise mine the air was sampled again two months later, 16 days after the mine had shut down.

PARADISE MINE.

The Paradise mine is on the railroad 3 miles east of Duquoin. The depth at the shaft was 380 feet and the production was about 1,800 tons a day with a force of 275 to 280 miners. The workings were mainly to the north of the shaft. The strata dip east with an inclination of $1\frac{1}{2}^{\circ}$. One thousand four hundred and fifty feet west of the shaft is a fault in which the beds on the west side drop 6 feet. The faulted block, which is about 700 feet wide, extends west to another fault with a displacement of 30 feet also on the west side. This latter block is flexed a short distance west of the fault by a sharp upturn on which the beds rise rapidly to the west. A short distance northwest of the shaft there is a marked basin, and then a rapid rise in which considerable gas was found. The coal bed is 10 feet thick, but in the first mining $7\frac{1}{2}$ feet is taken out. The largest amount of gas was noted in the newest workings that were being vigorously advanced, especially in one area one-fourth of a mile southeast of the shaft.

The return air in this mine was sampled on February 15 on the main haulage road and runaround just north of the shaft. The volume was 67,450 cubic feet a minute, and it contained 0.15 per cent methane, or 101 cubic feet a minute, and 0.16 per cent of carbon dioxide. On April 15, after the mine had been shut down for 16 days, the air was sampled at the same locality; it had a volume of 15,770 cubic feet a minute and contained 0.26 per cent methane, 0.91 per cent carbon dioxide, and 19.15 per cent oxygen. This is equivalent to 41 cubic feet of methane, or less than half the volume at the time the previous sample was taken.

A special sample was taken on the first entry extending southeast of the north main east return 465 feet east of the shaft. Six miners had been working in this entry, which extended down the slope of the beds just one-fourth of a mile, with two small chambers at its lower end. At the time of the sampling the mine had not been worked for two days. The air in this return had a volume of 5,530 cubic feet a minute and contained 0.16 per cent methane and 0.11 per cent carbon dioxide. On April 15, 16 days after mining had ceased, the velocity of the ventilation seemed to be considerably diminished, as only about 1,000 feet of air was passing. This air was found to contain 0.33 per cent methane and 0.19 per cent carbon dioxide. It was reported that in this mine on March 15 a blower had been opened in room 5 off the fifth south gangway off No. 1 east entry.

Some face samples taken by S. O. Andros, of the department of mining engineering of the University of Illinois, in rooms on the fifth southeast A entry, contained considerable methane. One from

room 5 contained 5.53 per cent and one from room 2, 10.35 per cent, showing that considerable gas was being given off in some of the working faces.

MAJESTIC MINE.

The Majestic mine is in the bottom of the basin 3 miles southeast of Duquoin and at the shaft the coal is about 420 feet below the surface. The ordinary production was 2,150 tons a day and about 300 miners were employed. The return air was sampled on February 15, after the mine had been idle two days. The main east return, which was sampled 100 feet east of the shaft, had a volume of 58,650 cubic feet a minute, and contained 0.14 per cent methane, or 82 cubic feet a minute, and 0.18 per cent carbon dioxide. The air in the main west return just west of the shaft had a volume of 68,530 feet and contained 0.12 per cent methane, or 83 cubic feet a minute, and 0.14 per cent carbon dioxide. The air in each of these returns ventilated workings about equal in extent and in each the number of miners employed was about the same. The uniformity in volume of methane shows that the conditions under which the gas was given off must have been uniform throughout the workings.

DAVIS MINE.

The Davis mine is one-half mile southwest of the Paradise mine and upon a slope of the monocline on which the strata rise rapidly toward Duquoin. The depth at the shaft is 320 feet and the production is 1,400 tons a day. Gas is noticed at places but no large amounts have been encountered. The mine was inspected on February 15, after mining had been suspended for two days. The workings are north of the shaft; the air is about equally divided in two currents between the west-side and the east-side workings. The main east return was found to carry 44,550 cubic feet of air a minute, containing 0.16 per cent methane, or 71 cubic feet a minute, and 0.27 per cent carbon dioxide. The west return carried 31,080 feet, containing 0.08 per cent methane, or 25 cubic feet a minute, and 0.10 per cent of carbon dioxide. The fact that nearly three times as great a volume of methane was carried in the east return as in the west return could not be accounted for. Doubtless the coal is more gaseous in the east side than in the west.

HORN MINE.

The Horn mine is 2 miles southwest of Duquoin. The coal is $7\frac{1}{2}$ feet thick and at the shaft is about 80 feet below the surface. The return air was sampled on February 15 and found to carry no methane. The return air from the south half of the mine had a volume of about 7,200 cubic feet a minute and contained 0.19 per cent carbon dioxide;

the air in the north return had a volume of 14,800 feet and contained 0.36 per cent carbon dioxide. These results probably indicate that all of the methane had escaped, but considerable carbon dioxide was being produced by oxidation. At the time of sampling the mine was not being worked extensively.

MINES NEAR WEST FRANKFORT.

In several of the mines in the vicinity of West Frankfort the operators have had considerable difficulty with gas, and the belief is current that this district is the most gaseous one in the southern Illinois field. There have been explosions in the Dering No. 18 mine, one of which resulted in a serious fire, and at various times men have been injured to a greater or less extent by igniting gas. The mines are relatively deep, ranging from 440 feet at the Wilmington-Star to 512 feet at the Dering No. 18. The measures dip at a very low angle to the north or slightly northeast, the average slope being nearly 25 feet to the mile. There are no notable disturbances of the strata in the area, except occasional low rolls. The coal is the Herrin bed, which is about 8 feet thick at the Wilmington-Star mine and about 10 feet at the Dering No. 18.

DERING NO. 11 MINE.

The Dering No. 11 mine, which is about a mile north of West Frankfort, has a production of about 1,500 tons a day. The return air was sampled at several points in the mine, and also in the total upcast in the hoisting shaft on January 19, when the mine was in full operation, and again on April 4 after it had been shut down four and one-half days. The principal results are given in the table on page 189. The volume of the total return air, 92,820 cubic feet per minute, was obtained by the addition of the volumes of the separate splits in the mine. The analyses indicate that the total emanation of methane on January 19 was 204 cubic feet a minute. The sample from the third north return, in which a sample was taken 500 feet northeast of the shaft, represented air from the third and fourth northeast sections of the mine which were not being worked at the time of sampling but had been in operation the day before. The return air from the seventh and eighth north main entries and east fifth, sixth, seventh, and eighth southeast entries was taken 600 feet northeast of the shaft. This return carried about two-thirds of the air from the east side, and together with the third north return carried the entire ventilation from the east side of the mine, where about 125 miners were working ordinarily. This return carried 82 cubic feet of methane a minute. The air in the first northwest return was sampled 400 feet northwest of the shaft; it carried 33 cubic feet of methane a minute. It drained workings in which there had been a squeeze some time previously. The workings had since been

opened, although not yet mined. The air of the main west return was found to carry 83 cubic feet of methane a minute or practically the same as in the main east. This air ventilated all the southwest and western part of the mine where 45 miners were employed in workings which were less than half as extensive as those on the east side. Another series of samples was taken on March 28 at the same places, and the sampling was repeated on April 4 after the mine had been shut down for four and one-half days. The results of the three series are given in the following:

Methane and carbon dioxide in return air of Dering No. 11 mine, Jan. 19, Mar. 28, and Apr. 4, 1912.

Return.	Jan. 19.				Mar. 28.				Apr. 4.			
	Volume of air per minute.	Methane content.	Carbon dioxide content.	Volume of methane per minute.	Volume of air per minute.	Methane content.	Carbon dioxide content.	Volume of methane per minute.	Volume of air per minute.	Methane content.	Carbon dioxide content.	Volume of methane per minute.
Total upcast in main shaft...	<i>Cu. ft.</i> 92,820	<i>P. ct.</i> 0.22	<i>P. ct.</i> 0.11	<i>Cu. ft.</i> 204	<i>Cu. ft.</i> 84,625	<i>P. ct.</i> 0.32	<i>P. ct.</i> 0.13	<i>Cu. ft.</i> 271	<i>Cu. ft.</i> 71,068	<i>P. ct.</i> 0.25	<i>P. ct.</i> 0.10	<i>Cu. ft.</i> 177
Main west.....	25,200	.29	.12	83	18,914	.48	.11	91	15,850	.36	.07	57
First north-west a.....	22,050	.15	.12	33	21,295	.15	.13	32	19,656	.14	.23	27½
Main east b.....	37,290	.22	.08	82	35,420	.19	.13	67	28,182	.24	.06	68
Third north-east b.....	8,280	.25	.20	21	9,288	.26	.12	24	7,380	.28	.21	20½

a Workings not being mined.

b Total east return in these two returns.

The volume of methane in the total return and in the different returns was remarkably constant in January and March, but a general diminution was shown on April 4, four days after mining had ceased. Duplicate samples of the upcast taken by S. O. Andros in the hoisting shaft on January 9, 1912, were found to contain 0.27 and 0.29 per cent methane and 0.06 and 0.03 per cent carbon dioxide, respectively. The total volume of return air was estimated by Andros at 120,000 feet, but as it was much less than this at the time of the author's visits it probably may be taken at 90,000. This gives about 250 cubic feet as the total volume of methane per minute, a figure close to the average of the volumes found under similar conditions on January 19 and March 28 of the same year. Two entry samples were taken in duplicate by Andros at the same time; one taken at the face of the sixth stub entry off the fourth southeast entry contained 0.19 and 0.48 per cent methane, and another collected in the fifth stub off the same entry contained 0.16 and 0.05 per cent methane. The air in the latter return came from the third, fourth, fifth, and sixth panels and ventilated 30 rooms. The volume was not determined; the carbon dioxide content ranged from 0.02 to 0.06 per cent.

DERING NO. 18 MINE.

Dering No. 18 mine, situated 3 miles northeast of Dering No. 11 mine is in somewhat deeper-lying coal as the beds dip in that direction, but otherwise the conditions are similar to those in No. 11. The coal is 512 feet below the surface at the shaft and the bed is about 10 feet thick. The output is about 500 tons a day. Considerable amounts of gas were apparent at intervals in the mine, especially on the north side, where it could be ignited in some of the upgrade workings. The air in the returns was measured and sampled January 19, when the mine was working most of the time, and again on April 4, after it had been shut down four and one-half days. The principal results are given in the following table:

Methane and carbon dioxide in return air of Dering No. 18 mine, Jan. 19 and Apr. 4, 1912.

Place where sample was taken.	Jan. 19.				Apr. 4.			
	Volume of air per minute.	Methane content.	Carbon dioxide content.	Volume of methane per minute.	Volume of air per minute.	Methane content.	Carbon dioxide content.	Volume of methane per minute.
Main south return, 350 feet south-east of shaft.....	<i>Cu. ft.</i> 18,240	<i>P. ct.</i> 0.06	<i>P. ct.</i> 0.05	<i>Cu. ft.</i> 11	<i>Cu. ft.</i> 13,500	<i>P. ct.</i> 0.11	<i>P. ct.</i> 0.05	<i>Cu. ft.</i> 15
West return off south side.....	9,180	.21	.05	19	4,000	.25	.05	10
Main north return.....	24,000	.08	.05	19	20,790	.06	.04	12½
Total.....	51,420			49	38,290			37½

There were working places for about 60 miners on the south side and the same number on the north side, but no mining was being done on the north side on January 19 or on the west return off the south side. The larger amount of methane from the south workings indicates a greater emanation, and owing to this variation the total amount on the south side (30 cubic feet) was considerably greater than the volume from the north side. Except in the main south return, there was a marked diminution in methane on April 4; but at this time a small part of the air from both returns, 6,000 cubic feet in all, made a short circuit to the upcast. Possibly the volume of methane in this leak was 4 cubic feet a minute.

WILMINGTON-STAR MINE.

The Wilmington-Star mine, which is one-fourth of a mile south of West Frankfort, is a new mine actively worked. The coal is 440 feet below the surface at the shaft and the bed is 8 feet thick. The mine is somewhat gaseous, but not more so in one part than in another, and occasional feeders are found. All the drill holes in the coal are gaseous and in most of them the gas can be lighted. The ventilation

comes to the upcast in two returns, one from the south half and the other from the north half of the mine. The air in these returns was sampled on January 19 and March 28 while the mine was in active operation, producing about 1,500 tons a day, and again on April 11, when mining had been suspended 12 days. The results are given in the following table:

Methane and carbon dioxide in return air of Wilmington-Star mine, January to April, 1912.

Return.	Jan. 19.				Mar. 28.				Apr. 11.			
	Volume of air per minute.	Methane content.	Carbon dioxide content.	Volume of methane per minute.	Volume of air per minute.	Methane content.	Carbon dioxide content.	Volume of methane per minute.	Volume of air per minute.	Methane content.	Carbon dioxide content.	Volume of methane per minute.
Main north.....	<i>Cu. ft.</i> 61,800	<i>P. ct.</i> 0.14	<i>P. ct.</i> 0.05	<i>Cu. ft.</i> 86½	<i>Cu. ft.</i> 70,070	<i>P. ct.</i> 0.17	<i>P. ct.</i> 0.06	<i>Cu. ft.</i> 119	<i>Cu. ft.</i> 22,400	<i>P. ct.</i> 0.31	<i>P. ct.</i> 0.06	<i>Cu. ft.</i> 69
Main south.....	61,067	.14	.09	85½	58,240	.16	.72	93	25,410	.24	.04	61

The air of the main north return, which was from all workings north of the shaft, was sampled 300 yards north of the shaft. Sixty-two miners were employed in the north workings. The sample of south-return air was taken 800 feet southwest of the shaft. On March 28 the methane had increased since the previous test to 93 cubic feet in the main north return and 119 cubic feet in the main south return, and there was a notable increase in carbon dioxide on the south side. On April 11, after 12 days of no mining, the air volume was diminished to less than half. The volume of methane was 61 cubic feet a minute from the south side and 69 cubic feet from the north side, a marked diminution which can be clearly connected with the cessation of mining.

MINES ABOUT JOHNSTON CITY.

The coal of the Herrin bed is mined extensively about Johnston City. The depth of the coal bed below the surface in these mines varies from 96 to 276 feet, increasing gradually from south to north.

STANDARD OR EAST MINE.

The deepest mine is the Standard or East mine a mile southeast of Johnston City, where the coal at the shaft lies 276 feet below the surface. The thickness of the coal worked is about 7 feet. The air was measured and sampled on February 7, when the mine was in full operation, and again on April 11, 12 days after mining had ceased. Considerable methane appeared in places, especially in the region

drained by the main west return, and at times small explosions have occurred. The air in the east return, which was sampled 450 feet southeast of the shaft, ventilated workings in which 65 miners were employed. Its volume was 14,100 cubic feet and it contained 0.26 per cent methane, or nearly 37 cubic feet a minute, and 10 per cent carbon dioxide.

The air from the western half of the mine, in which 65 miners were employed, was sampled in the first northwest return 450 feet southwest of the shaft; it had a volume of 15,620 cubic feet a minute and contained 0.12 per cent methane, or 19 cubic feet a minute, and 0.09 per cent carbon dioxide. A small split of the main west-return air in the main west gangway, 250 feet south and 1,350 feet west of the shaft, was also sampled. It had a low velocity and contained 0.43 per cent methane and 0.32 per cent carbon dioxide.

On April 11, after 12 days of no mining, the east return 200 feet east of the shaft was found to have a volume of 10,950 cubic feet of air a minute, containing 0.29 per cent methane, or 32 cubic feet a minute, and 0.06 per cent carbon dioxide. The volume of the west return was 11,970 feet, containing 0.25 per cent methane, or 30 cubic feet a minute, and 0.07 per cent of carbon dioxide. At the first visit the upcast was to the fan, while on the later one the upcast was at the hoisting shaft, and the fan was moving at about half speed, but otherwise the air courses were unchanged. For some time past the main west entry had been very gaseous, so that it had to be bratticed up to the face. However, the amount of methane on that side was practically the same on April 11 as on February 7.

WEST MINE.

At the West mine, on the western edge of Johnston City, the coal is 220 feet below the surface at the shaft. The bed is $8\frac{1}{2}$ to 9 feet thick. The mine was new but produced about 2,800 tons of coal a day. Its ventilation was divided into an eastern half and a western half, there being about 200 miners on each side. On February 7 the main east return 150 feet from the shaft was found to carry 33,920 cubic feet of air per minute, which contained 0.28 per cent methane, or 95 cubic feet a minute, and 0.25 per cent carbon dioxide. The main west return 150 feet northwest of the shaft carried 24,500 cubic feet of air, which contained 0.04 per cent methane, or 10 cubic feet a minute, and 0.07 per cent of carbon dioxide. Uniform amounts of methane were reported to be given off in the working faces in the mine and many holes could be lighted, but no blowers had been observed. The much greater volume of methane in the east return as compared with that in the west return, nearly 10 to 1, has no apparent explanation.

BLACKBRIER MINE.

The Blackbrier mine, a mile south of Johnston City is 185 feet deep at the shaft and produces 1,100 tons a day; 156 miners are employed. The active workings in this mine are all east of the shaft. The return air from them was sampled on March 12, and again on April 11, 12 days after mining had ceased. On March 12 the return air 150 feet east of the shaft had a volume of 23,120 cubic feet and contained 0.24 per cent methane, or 55½ cubic feet a minute, and 0.46 per cent carbon dioxide. The air in the same return taken at the 11th south entry a quarter of a mile farther in, had a volume of 9,240 cubic feet a minute and contained 0.30 per cent methane and 0.30 per cent carbon dioxide. One hundred and twenty miners were employed in these workings. These same currents were tested on April 11. At the place 150 feet east of the shaft, the return had a volume of 22,000 cubic feet of air containing 0.14 per cent methane, or 31 cubic feet a minute, and 0.28 per cent carbon dioxide. The volume of methane given off in this mine after mining had ceased diminished from 55½ to 31 cubic feet, a marked decrease.

When the shaft was sunk in 1894 a great gas blower was opened in driving the main entry, 20 feet west of the shaft. The gas was drained off in a 4-inch pipe and burned; it sustained a large flame for two years. When the entry reached a point 150 feet east and 800 feet south of the shaft, another blower was opened, causing an explosion that killed five men. Soon after this blower began the one near the shaft bottom ceased. Finally the second one also died out. The old workings west of the shaft were never very gaseous, most of the gas occurring in working faces to the extreme northeast.

NEW VIRGINIA MINE.

The New Virginia mine is 1½ miles south of Johnston City and nearly a mile south of the Blackbrier mine. It produced 1,100 tons a day and about 125 miners were employed. The coal is about 6½ feet thick and lies 120 feet below the surface at the shaft. Very little gas has been observed in the mine and it was mostly in the northwest workings. The air in the mine was sampled on March 12. The main return carrying all of the south and part of the west ventilation, from workings in which 22 miners were employed, was tested 150 feet southwest of the shaft. This return air was found to have a volume of 10,170 cubic feet a minute and it contained 0.06 per cent of methane, or 6 cubic feet a minute, and 0.23 per cent of carbon dioxide. The north return carried air from part of the west workings and a large area of old east workings where 38 to 40 miners were working. The return air, which was sampled 150 feet northwest of the shaft where

the volume was 10,908 cubic feet a minute, contained 0.06 per cent methane, or $6\frac{1}{2}$ cubic feet a minute, and 0.33 per cent carbon dioxide. The southeast return carried the balance of the air from the east and south workings, in which 60 miners were employed. This return was near the upcast 100 feet north of the shaft where the volume of the passing air was found to be 15,120 cubic feet of air a minute. The air contained 0.02 per cent methane, or 3 cubic feet a minute, and a large proportion of carbon dioxide, 1.43 per cent.

CARTERVILLE DISTRICT MINE.

The Carterville District mine, a half mile south of the New Virginia mine, is a somewhat larger producer than the others, the production being 1,450 tons. It is only 96 feet deep at the shaft. At the time of the author's visit on March 12, the fan was running at about half speed. The air of the main west return which ventilated the west half of the mine, in which 77 miners were employed, was sampled 370 feet west of the shaft. The volume of air was 7,250 feet and it contained 0.16 per cent carbon dioxide and no methane. In the main east return, from workings in which about 48 miners were employed, 350 feet east of the shaft the volume of air was 10,530 cubic feet and it contained 0.02 per cent methane, or 3 cubic feet a minute, and 0.30 per cent carbon dioxide.

OAK RIDGE MINE.

At the Oak Ridge mine, $1\frac{1}{2}$ miles west of Johnston City, the conditions were closely similar to those in the Blackbrier mine a mile east. The depth is 135 feet at the shaft, the daily production was 1,000 to 1,200 tons, and 145 miners were employed. The coal, which is about 9 feet thick, gave off considerable gas in places, mostly on the north side, where the gas could be lighted in drill holes and crevices in the faces. There is a 40-foot fault with a drop on the west side along the west margin of the workings and a smaller fault northeast of the shaft, but these were not gaseous. The greater part of the return air from this mine was measured and sampled at a point 240 feet west of the shaft. This air also ventilated a large area of old workings. Its volume on March 25 was 20,880 cubic feet a minute and it contained 0.08 per cent of methane, or 16 cubic feet a minute, and 0.18 per cent of carbon dioxide. The east-return air 100 feet east of the shaft had a volume of 25,500 cubic feet, and it contained 0.01 per cent methane, or $2\frac{1}{2}$ cubic feet a minute, and 0.07 per cent carbon dioxide. The total volume of methane was 25 cubic feet a minute or just about the same as in the Colp mine, which was visited later.

MINES NORTHEAST OF MARION.

There are several mines in the Herrin coal bed between Marion and Pittsburgh. In this region the coal measures dip to the northeast at low angles and there are no special disturbances of the strata.

COLP MINE.

The Colp mine, which is a mile northwest of Pittsburgh, is a small, new mine with a daily output of about 400 tons. Thirty-six miners were employed at the time of the author's visit. The coal bed is $5\frac{1}{2}$ to 6 feet thick and lies 210 feet below the surface. The total return air, which was sampled on March 26 near the shaft, had a volume of 40,600 cubic feet, and it contained 0.06 per cent methane, or 24 cubic feet per minute, and 0.07 per cent carbon dioxide.

SCRANTON MINE.

The Scranton mine, a mile southwest of the Colp mine, was 160 feet deep at the shaft. At the time of the author's visit 90 miners were employed, and the production was about 650 tons of coal a day. The total return air, sampled March 26 at two places near the shaft, had a volume of 31,500 cubic feet a minute, and it contained 0.08 per cent methane and 0.08 per cent carbon dioxide.

WEST VIRGINIA MINE.

The West Virginia mine, which is a mile southwest of the Scranton mine, or $2\frac{1}{2}$ miles northeast of Marion, is somewhat larger than the other two, its output being 1,200 tons a day. The depth at the shaft is 108 feet and the coal is 6 to $7\frac{1}{2}$ feet thick. Some gas was observed in places in this mine, especially a small feeder, and in many of the drill holes it could be lighted. The return air was sampled on March 26. The main north return near the shaft carried 26,112 feet of air, containing 0.01 per cent of methane, or $2\frac{1}{2}$ cubic feet a minute, and 0.08 per cent of carbon dioxide. There was a small admixture of air from the south part of the mine in this return in a runaround near the shaft. Fifteen miners were employed in the workings ventilated by this air current. The air in the south return, which ventilated about two-thirds of the mine and supplied 100 miners, was sampled at the shaft. Its volume was 36,260 cubic feet, and it contained 0.09 per cent carbon dioxide and no methane.

MINES NORTHWEST OF MARION.

There is a group of mines 3 miles northwest of Marion in which the conditions present some interesting features. Three were examined, the Peabody No. 2 and No. 3 and the Big Muddy.

PEABODY NO. 3 MINE.

At the Peabody No. 3 mine the coal is 100 feet below the surface at the shaft and although the bed is 9 feet thick only $7\frac{1}{2}$ feet is taken out. It is covered by a thick body of shale and the strata dip northwest at a low angle. The mine produces 2,400 tons a day. The ventilation is in two main airways, both draining about the same extent of work-

ings, but there is some mingling of air currents in certain parts of the mine. The main southwest and southeast returns combined, of which a sample was taken about 300 feet south of the shaft, were found to carry 24,651 feet of air, which contained 0.03 per cent methane, or 7 cubic feet a minute, and 0.21 per cent of carbon dioxide. This return comes from workings in which 120 miners were employed. The main north-return airway was found to carry 12,320 feet of air, which contained 0.06 per cent of methane, or 7 cubic feet a minute, and 0.23 per cent carbon dioxide. About 80 men were mining coal in the workings ventilated by this air. The notable feature in these results is that equal volumes of methane were coming from the two halves of the mine, although the fresh workings in the deeper parts were somewhat less extensive than those on the south side. The small total volume is undoubtedly due to the shallowness of the mine and probably would be less were it not for the heavy body of shale overlying the coal.

PEABODY NO. 2 MINE.

A mile south of the Peabody No. 3 mine is the Peabody No. 2 mine, where the shaft reaches the Herrin coal at a depth of only 80 feet about a mile from the outcrop. This mine employed about 160 miners and produced 1,200 tons a day. The bed is 9 feet thick, but only 7½ feet was mined. The mine was about 500 acres in extent, most of the part lying east of the shaft being old workings. All the new workings were to the west. Some of the return air from this mine escaped to the south through outcrop workings. The upcast air at the shaft was found to contain 0.11 per cent methane and 0.11 per cent carbon dioxide. The volume was not determined but was stated to be about 50,000 cubic feet a minute, which would indicate 55 cubic feet of methane a minute, a volume greatly in excess of that from other shallow mines in the district. The main west and north returns in the new workings, where about 100 miners were employed, in which samples were taken 1,200 feet west of the shaft, carried 25,550 feet of air, which contained 0.05 per cent methane, or 13 cubic feet a minute, and 0.10 per cent carbon dioxide.

BIG MUDDY MINE.

The Big Muddy mine is a mile northwest of the Peabody No. 3 mine on the opposite slope of a low anticline. The strata dip westward. The shaft was 100 feet deep and the bed is 9 feet thick, of which 7 feet was taken out. The production was 1,500 tons a day.

No gas was ever observed by the miners in this mine. A fault with a throw of 51 feet extends from north to south across the east side of the mine and there are many rolls and minor faults.

The ventilating current in this mine came to the upcast in two main returns, one from the east half of the mine where 65 miners were working, and the other from the west section where 97 miners were working. The air was sampled on January 24, and again on April 13, two weeks after mining had been suspended.

In the first visit, the east return, at the head of the rock slope just east of the shaft, carried 28,620 cubic feet of air per minute which contained 0.04 per cent methane, or $11\frac{1}{2}$ cubic feet a minute, and 0.10 per cent carbon dioxide. The west return, in the second south, sixth west entry, carried 5,700 feet of air which contained 0.05 per cent methane or 3 cubic feet a minute, and 0.14 per cent carbon dioxide. The small volume of the air was stated to be due to some break far out in the workings. The normal volume was about four times as much.

On April 13 the fan was running at greatly diminished speed, as mining was suspended. The east return, tested at the same place as before, was found to carry 14,668 feet of air which contained 0.02 per cent methane, equivalent to nearly 3 cubic feet per minute, and 0.19 per cent carbon dioxide. The air in the east return, one-half mile south of the shaft, had a slight velocity, and seemingly its volume was not over 1,000 feet a minute. It contained 0.02 per cent methane and 0.62 per cent carbon dioxide. These figures show a marked diminution in the amount of methane given off during the cessation of working and precisely the same volume of carbon dioxide in the east return and approximately the same in the west return.

MINES SOUTH AND WEST OF HERRIN.

Samples of return air were collected in the Hemlock, Taylor No. 1, and Taylor No. 2 mines south of Herrin. They were taken at two periods, one on March 8 when the mines were operating, and the others on April 10, 11 days after the mining had ceased.

The mines were relatively shallow, the Hemlock being 155 feet deep at the shaft, the Taylor No. 2, 140 feet, and the Taylor No. 1, 53 feet. The Hemlock mine is in the southern edge of the town; the Taylor No. 2 a half mile due south of the Hemlock mine; and the Taylor No. 1 about a mile due east of the Taylor No. 2. The mines were all small and relatively new with little old workings and no noteworthy structural features. The strata dip gently northward.

HEMLOCK MINE.

In both samplings of the Hemlock returns, the east and west return air currents were sampled separately. On March 8 the air in the main west return, at a point 250 feet west of the shaft, was found to have

a volume of 30,512 cubic feet a minute, and contained 0.02 per cent methane and 0.10 per cent carbon dioxide. The air in the east return, sampled 100 feet east of the shaft, had a volume of 23,870 cubic feet a minute and contained 0.01 per cent methane and 0.05 per cent carbon dioxide. The mine was being worked by 45 miners and its average daily output was 700 tons. The bed is 9 feet thick and about $7\frac{1}{2}$ feet of it was mined.

Little gas was reported in the mine but some methane was noticeable in the fresh headings in the downward pitch on the second north entry off the east gangway. The analyses indicate that the total outflow of methane from the Hemlock mine was only $9\frac{1}{2}$ cubic feet a minute, of which 7 cubic feet was from the west workings and nearly $2\frac{1}{2}$ cubic feet from the east workings.

On April 10, 11 days after the mine had ceased operating, the sampling was repeated. At this time the direction of the air current was reversed, the fan exhausting and running at much less speed. The air of the main east return, sampled near the foot of the fan shaft, had a volume of 19,180 cubic feet a minute and contained 0.04 per cent methane, or 8 cubic feet a minute, and 0.07 per cent carbon dioxide. The air of the main west return at the foot of the fan shaft had a volume of 17,892 feet, and contained about the same volume of methane as the east return, or a little less than 8 feet a minute, and 0.12 per cent of carbon dioxide. The volume of methane per minute in the west return was therefore slightly more than when the mine was in operation, whereas in the east return it was $2\frac{1}{2}$ times as much as in the first test, an increase that is difficult to explain.

TAYLOR NO. 2 MINE.

The conditions at the Taylor No. 2 mine were closely similar to those at the Hemlock, but the former is about twice as large. On March 8 the main west return carried 31,875 feet of air, which contained only 0.02 per cent methane, and the main east return and its parallel entry, about 26,832 feet of air, contained the same percentage; the carbon dioxide content was 0.09 per cent. There were at the time 80 miners on the west side and 100 on the east side, the entire output being about 1,500 tons a day. The total volume of methane was nearly $5\frac{1}{2}$ cubic feet a minute on the east side and nearly $6\frac{1}{2}$ cubic feet on the west side. The east side workings extended much deeper than those in the west side. On April 10, after 11 days of no working, the air volume was about half and no trace of methane was found in either return. The percentage of carbon dioxide was slightly increased, but the volume per minute was considerably less. No gas has ever been noticed in this mine.

TAYLOR NO. 1 MINE.

The Taylor No. 1 mine was only 55 feet deep and no gas had ever been reported in it. The production was 1,000 tons a day. The workings were equally divided into two sides, east and west, with about 150 miners in each side. The east return carried 14,000 cubic feet a minute, which contained only 0.01 per cent methane and 0.07 per cent carbon dioxide. The volume of air in the west return was 18,000 cubic feet a minute and the air contained 0.10 per cent carbon dioxide and no methane. The total volume of methane from the mine, therefore, was only 1.4 cubic feet a minute, and this was all from the east side where the workings were shallower than on the other side.

SUNNYSIDE MINE.

The Sunnyside mine, which is a mile west of Herrin, presented conditions similar to those in the Hemlock mine, but it was considerably larger. The depth to the coal was 150 feet at the shaft, $7\frac{1}{2}$ feet of coal was taken out, and the output was 2,000 tons a day, 200 miners being employed. The ventilation of the mine was by two main returns, the air of which was measured and sampled on January 31. The east return, 250 feet southwest of the shaft, carried 39,150 cubic feet of air, which contained 0.06 per cent methane, or $23\frac{1}{2}$ cubic feet a minute, and 0.03 per cent carbon dioxide. This air ventilated all the east side of the mine. The west return, 300 feet southwest of the shaft, was found to carry 47,554 feet of air, which contained 0.04 per cent methane, or 19 cubic feet a minute, and 0.03 per cent carbon dioxide. The similarity of the figures show closely uniform emanation of methane on the two sides of this mine.

CHICAGO & CARTERVILLE CO. MINES.

The "A" mine of the Chicago & Carterville Co. is 1 mile northwest of Herrin and the "B" mine 2 miles northeast of that city. The "A" mine was 186 feet deep and the "B" mine 247 feet. Both are mines of large extent and with old workings of considerable area. The "A" mine had an output of about 2,100 tons a day and the "B" mine had an output of 1,500 tons a day. The coal is $7\frac{1}{2}$ feet thick in the "B" mine and the strata dip gently with no especial local disturbances. The "B" mine is in a broad irregular syncline in the general northerly monocline with the dips northwest. Both of these mines were sampled while in full operation and again after operation had been suspended a while.

"A" MINE.

The main returns of the "A" mine came to the upcast in five principal courses, three of them from the north and two from the south. The proportions of methane and carbon dioxide on January 31 and April 10, 1912, were as follows:

Methane and carbon dioxide in return air from "A" mine, January and April, 1912.

Return.	Jan. 31.				Apr. 10.			
	Volume of air per minute.	Proportion of methane in return air.	Proportion of carbon dioxide in return air.	Volume of methane per minute.	Volume of air per minute.	Proportion of methane in return air.	Proportion of carbon dioxide in return air.	Volume of methane per minute.
Main north, 500 feet northwest of shaft.	<i>Cu. ft.</i> 33,120	<i>Per ct.</i> 0.25	<i>Per ct.</i>	<i>Cu. ft.</i> 83	<i>Cu. ft.</i> 14,000	<i>Per ct.</i> 0.15	<i>Per ct.</i> 0.12	<i>Cu. ft.</i> 21
Third west north, 500 feet north of shaft.....	15,576	.03	0.05	8	13,650	.10	.16	14
First east north.....	16,884	.20	1.21	33	32,680	.16	.14	52
Second west south.....	18,400	.19	1.36	35	13,860	.20	.61	27
Main south.....	31,668				20,800	.07	.11	14½

These figures show great difference in volume of methane given off in different parts of the mine, especially in the smaller amount in the third west north return, where there was a large coal production.

The sampling was repeated at the same places on April 10, 11 days after mining had ceased, but owing to a change in the airways some of the air of the returns which formerly came out at the main north return then came out by the first east north return after draining old workings. The total volume of methane in the north workings, however, was only 87 cubic feet a minute on April 10, as compared with 124 feet when the mine was in operation, and the west south return showed a slight diminution.

Disregarding the main south return the total volume of methane in the other returns was 159 cubic feet per minute on January 31 and 114 feet on April 10, a notable diminution. The increased amount in the first east north return was due mainly to the fact that the air volume was doubled and gas brought in from old workings on the main north return. The cause of the increase in the third west north return was not apparent.

In one part of the "A" mine an area of 325 by 500 feet had been walled off on account of defective ventilation. The air in this chamber contained a large volume of methane, and showed 0.35 inch of pressure on the water gage.

"B" MINE.

The "B" mine was visited on March 8 and April 12, 1912. The returns from this mine were in two courses, the main east ventilating workings operated by 75 miners, and the main west from workings

in which 150 miners were employed. On March 8 the main east return air 300 feet southeast of the shaft had a volume of 31,350 cubic feet per minute and contained 0.15 per cent methane and 0.05 per cent carbon dioxide. The air in the main west return 300 feet southwest of the shaft had a volume of 54,225 feet and contained 0.10 per cent methane and 0.06 per cent carbon dioxide. The volume of methane a minute was 47 cubic feet in the east return and 54 in the west return, a rather slight difference when it is considered that twice as much coal was being mined on the west side. On April 12, after 13 days of no mining, the air in the returns was measured and sampled at the same places. The fan was making 25 per cent fewer revolutions per minute. The main east return carried 26,250 feet of air containing 0.16 per cent methane and 0.11 per cent carbon dioxide, and the main west return carried 51,440 feet of air containing 0.19 per cent methane and 0.14 per cent carbon dioxide. The increase in the carbon dioxide is a notable feature. The total volume of the methane in the main east return had diminished slightly to 42 cubic feet a minute, whereas in the main west return it had increased to 98 feet, nearly double the volume on March 28.

POND CREEK, POSSUM RIDGE, AND REND NO. 2 MINES.

The Pond Creek mine, which is $3\frac{1}{2}$ miles north of Herrin, the Possum Ridge mine, a mile farther northeast, and the Rend mine to the northwest represent a part of the Herrin coal field in which there are only a few mines. The beds are on the regular north and northeast slope of the general monocline and present no notable local disturbances except a small fault in the northeast part of the Possum Ridge workings.

POND CREEK MINE.

The Pond Creek mine produces about 1,000 tons a day. The coal worked is about $7\frac{1}{2}$ feet thick and the depth at the shaft is 240 feet. There were two main-return air currents, which were sampled separately on March 18. The air of the main south return, which was sampled 300 feet southeast of the shaft, ventilated workings in which 57 miners were removing coal. The volume of air passing was 9,108 cubic feet per minute, and it contained 0.30 per cent methane, or 27 cubic feet per minute, and 0.11 per cent carbon dioxide. The air in the main west return, which was sampled 300 feet west of the shaft, ventilated workings in which 48 miners were employed. It carried 10,340 cubic feet of air, which contained 0.09 per cent methane, or 9 cubic feet a minute, and 0.15 per cent carbon dioxide. No reason is apparent for there being three times as much methane on the east side as on the west, in which the workings were only slightly more extensive. Gas shows occasionally in lamps in this mine, but its volume is seldom appreciable.

POSSUM RIDGE MINE.

The Possum Ridge mine is similar in production and extent of workings to the Pond Creek mine but the coal lies deeper, being 357 feet below surface at the shaft, and is $9\frac{1}{2}$ to 10 feet thick. The returns are from east and west sides with about two-thirds of the workings on the west side.

On March 18, the main west return 350 feet southwest of the shaft was found to carry 31,724 cubic feet of air per minute, which contained 1.19 per cent methane and 0.20 per cent carbon dioxide. The volume of methane in this return was $377\frac{1}{2}$ cubic feet a minute, which is the highest found in any of the mines visited in this district. The cause of the great volume of methane from the west workings was not apparent and its existence there was not known to the operators. Sixty-five miners were working on the west side.

The main east return 350 feet southeast of the shaft carried 20,237 feet of air which contained 0.07 per cent methane or 14 cubic feet a minute, and 0.08 per cent carbon dioxide. Thirty miners worked on the east side.

REND NO. 2 MINE.

The Rend No. 2 mine presented conditions closely similar to those in the other two mines. The depth at the shaft was 215 feet, $7\frac{1}{2}$ to 8 feet of coal was removed, the output was 2,000 tons a day, and 175 miners were employed. There were two main returns, one from the west and one from the east workings, but there was much admixture of air back in the mine. On March 18, the main west return just west of the shaft carried 59,160 cubic feet of air per minute, which contained 0.05 per cent methane, or $29\frac{1}{2}$ cubic feet a minute, and 0.06 per cent carbon dioxide; and the main east return, 20 feet east of the shaft, carried 47,600 feet of air containing 0.07 per cent methane or 33 cubic feet a minute, and 0.03 per cent carbon dioxide. An air current of about 14,000 feet comes to the shaft through a pump hole, about half from the east and half from the west. If it be assumed that this return carried an average of about 0.06 per cent methane or 8 cubic feet per minute the total methane in the returns was 71 cubic feet a minute. Gas was given off in large amounts in the south and east entries which are on an upgrade. Several small faults were somewhat gaseous, one of them especially so.

MINES NORTH OF CARTERVILLE.

There are large mines at frequent intervals north of Carterville following the Herrin coal down the north sloping monocline. In the first one, the Burr C, a mile out of town, the coal is at a depth of 100 feet, whereas the coal in the No. 8 mine of the Big Muddy Coal & Iron Co., which is 5 miles north, is 169 feet below the surface.

BURR C MINE.

At the Burr C mine, which had an output of 1,500 to 1,600 tons a day, the return was in two main currents, which were measured and sampled on March 5, 1912. The main east return 1,300 feet east of the shaft carried 11,200 cubic feet of air per minute, containing 0.06 per cent methane, or $6\frac{1}{2}$ cubic feet a minute, and 0.26 per cent carbon dioxide. This air ventilated the south half of the east side of the mine or about one-fourth of the area of active workings, in which 53 miners were employed. The return air from the main west and the north half of the east-side workings, which was sampled 1,000 feet east of the shaft, ventilated about three-fourths of the workings, in which 130 miners were employed. The volume was 47,500 cubic feet of air and it contained 0.01 per cent methane or $4\frac{1}{2}$ cubic feet a minute, and 0.11 per cent carbon dioxide. The smaller volume of methane in this return as compared with that from the area of smaller workings on the east side is a notable feature.

MADISON NO. 8 MINE.

The Madison No. 8 mine is a mile north of the Burr C mine. About 180 miners were employed, and the output averaged nearly 1,500 tons a day. The depth at the shaft was 100 feet. The air of the main return was sampled 10 feet west of the hoisting shaft and measured at several places in the vicinity. The total volume of upcast air was 96,517 cubic feet per minute, of which about 30,000 feet escaped through an old shaft. The return air at the shaft contained 0.01 per cent methane or $9\frac{1}{2}$ cubic feet a minute, and 0.15 per cent of carbon dioxide.

MADISON NO. 9 MINE.

The Madison No. 9 mine is 2 miles north of the No. 8 mine and the shaft was 120 feet deep. The mine was larger than the others, having an output of 2,500 tons a day. Seven and one-half feet of coal was mined. The main returns were in two airways, one from the west half of the mine, in which 200 miners were employed, and the other from the east half, in which 150 miners were employed. The main west return air was sampled 250 feet west of the shaft and carried 58,743 cubic feet of air, which contained 0.06 per cent methane, or 35 cubic feet a minute, and 0.09 per cent carbon dioxide. The east return, at the overcast 350 feet east and south of the shaft, carried 48,858 feet of air, which contained 0.07 per cent methane, or 34 cubic feet a minute, and 0.12 per cent carbon dioxide. The uniformity in amounts of methane on the two sides of the mine and the much greater amount as compared to that in the other mines south are noteworthy features.

Some gas was given off in the east-side workings and on the west side there was a feeder that burned continuously with a blaze 2 feet high. No gas was reported in drill holes. Most of the notable gas is in narrow zones of soft, wet coal extending across the mine and it appears to issue from the bottom clay.

C. & B. M. MINE.

The C. & B. M. mine of the Carterville & Big Muddy Co. is a short distance northeast of Cambria, or $1\frac{1}{2}$ miles northwest of the Burr C mine. The depth was 86 feet at the shaft and $7\frac{1}{2}$ to 8 feet of coal was mined from a 10-foot bed. The output was 900 to 1,100 tons, which was mined by about 150 miners. The return air from all but a very small part of the workings was sampled just below the south-west entry 1,800 feet north of the shaft. The return air had a volume of 36,000 feet and it contained 0.02 per cent methane, or 7 cubic feet a minute, and 0.22 per cent carbon dioxide. The air not represented in the sample was a current of about 4,000 cubic feet, ventilating workings in which 6 miners were taking down coal. The air passed into the upcast farther up the slope. No gas has ever been noted in this mine.

MINES NO. 7 AND NO. 8 OF THE BIG MUDDY COAL & IRON CO.

The two large mines of the Big Muddy Coal & Iron Co., mine No. 7, 1 mile east of Herrin, and mine No. 8, 3 miles northwest of Herrin, are in the Herrin coal bed.

Mine No. 8.—Mine No. 8 produces about 2,500 tons a day. The coal is at a depth of 170 feet near the shaft and $7\frac{1}{2}$ feet was mined in most of the workings. The return air came to the upcast in two main airways, one from the eastern part and the other from the western part of the mine, although there was a slight mingling of the ventilation far back in the workings. Little gas was found in the mine and most of it was in the top coal and never in the holes or undercuts. Some time previous to the date of sampling a small blower, which lasted two months, developed in the top in the second southeast entry. The east return, sampled at a point 195 feet east and 190 feet south of the shaft, carried 41,580 feet of air which contained 0.21 per cent methane, or slightly more than 87 cubic feet a minute, and 0.36 per cent carbon dioxide. The west return, sampled at a point 195 feet west and 190 feet south of the shaft, carried 36,190 feet of air which contained 0.14 per cent methane, or 51 cubic feet a minute, and 0.24 per cent carbon dioxide. The total volume of methane coming from this mine was about 139 cubic feet a minute.

Mine No. 7.—Mine No. 7, east of Herrin, was visited on March 18 and again on April 10, 11 days after mining had ceased. The output of this mine was about 2,000 tons a day. The coal is at a depth

of 150 feet near the shaft and about $7\frac{1}{2}$ feet was mined. The air in this mine was carried to the upcast by four principal airways, each one bringing the ventilation of about one-quarter of the mine. The tests of these returns gave the following results:

Methane and carbon dioxide in return-air samples collected in March and April, 1912, from mine No. 7.

Return.	Place where sample was taken.	Mar. 18, 1912.					Apr. 10, 1912.				
		Number of miners employed.	Volume of air per minute.	Proportion of methane in return air.	Proportion of carbon dioxide in return air.	Volume of methane per minute.	Volume of air per minute.	Proportion of methane in return air.	Proportion of carbon dioxide in return air.	Volume of methane per minute.	
Southeast, first to sixth south.	1,500 feet southeast of shaft.	55	<i>Cu. ft.</i> 21,680	<i>Per ct.</i> 0.13	<i>Per ct.</i> 0.70	<i>Cu. ft.</i> 6	<i>Cu. ft.</i> 2,700	<i>Per ct.</i> 0.12	<i>Per ct.</i> 0.80	<i>Cu. ft.</i> 3	
Southwest, first to sixth south.	1,500 feet northwest of shaft.	35	4,810	.06	1.45	3	525	.04	.57	$\frac{1}{2}$	
Northwest, second to sixth north and main west.	do	70	13,650	.18	.44	24½	11,050	.14	.36	15½	
Northeast, first to sixth northeast and main east.	1,500 feet southeast of shaft (at junction of northeast and main east).	70	16,240	.21	.36	34	8,085	.22	.38	18	
Total.....		230	56,380			68	22,360			37	

The variations in methane emanation from the four quarters of this mine are somewhat notable. Seemingly the south half was producing less gas than the north half, due allowance being made for the smaller extent of the fresh workings. These differences continued after the cessation of mining, when the total methane emanation diminished to nearly one-half. If the large area of the mine be taken into consideration the amount of methane is remarkably small, and if the relative area of coal surface were made the basis of comparison the mine would show probably less methane per square foot than other mines of the same depth.

BUSH AND ROYALTON MINES.

The mines at Bush and Royalton illustrated the conditions in the Herrin coal field in the northwest corner of Williamson County and the adjoining southwest corner of Franklin County. The strata dip northeast, but at Bush there is a local synclinal flexure in the general monocline.

Bush mine.—The Bush mine is a moderately large one, about 250 miners being employed and the daily production being 1,500 tons. The coal is 7 to 9 feet thick and lies 118 feet below the surface at the shaft. No appreciable amount of gas had ever been noted in the mine. The most extensive workings were on the east and north

sides. The main returns were east and west, but the currents from either side mixed somewhat far back in the mine. On March 25, 1912, the east return just east of the shaft carried 24,200 cubic feet of air per minute which contained 0.01 per cent methane and 0.22 per cent carbon dioxide. The main west return 350 feet west of the shaft carried 45,738 feet of air which contained 0.02 per cent methane and 0.34 per cent carbon dioxide. The volume of methane per minute was $2\frac{1}{2}$ feet on the east side and 9 feet on the west side.

Royalton mines.—The south and north mines at Royalton reach the coal at depths of 212 and 315 feet, respectively, the measures having a moderately steep dip to the north. The coal is 9 feet thick. The south mine was a small one with an output of 250 tons a day, and the north mine produced about 1,600 tons a day. Both mines were visited on March 25, 1912.

The total return of the south mine sampled near the shaft carried 41,395 feet of air which contained 0.03 per cent methane, or $12\frac{1}{2}$ cubic feet a minute, and 0.06 per cent carbon dioxide. No gas had ever been observed in making lamp tests in this mine. At the north mine a mile north, which was more than 100 feet deeper, gas appeared in the drill holes and in such volume at the working faces that bratticing had to be kept up close. The mine had been in operation only two years, so that there were no old workings. The returns were in two airways, each representing about half of the mine, and there were 90 miners on each side. The southwest return 100 feet southwest of the shaft carried 16,170 cubic feet of air per minute which contained 0.09 per cent methane, or $14\frac{1}{2}$ cubic feet a minute, and 0.21 per cent carbon dioxide. The southeast return, sampled 350 feet southeast of the shaft, carried 6,000 feet of air which contained 0.36 per cent methane, or $21\frac{1}{2}$ cubic feet a minute, and 0.10 per cent carbon dioxide. The reason for the greater volume of methane on the southeast side was not apparent.

HALLIDAYBORO MINE.

The Hallidayboro mine is far away from other mines in the Herrin bed. Its depth is 160 feet at the shaft and from 7 to 10 feet of coal was taken out. The output was about 1,200 tons a day. At the time of the author's visit on March 19, 1912, there were two main returns, one from the east, ventilating about three-fifths of the mine, and one from the west, ventilating two-fifths of the mine; 80 miners were employed in the east return and 45 in the west. The main east return 600 feet east of the shaft carried 23,100 cubic feet of air which contained 0.06 per cent methane, or 14 cubic feet a minute, and 0.16 per cent carbon dioxide. The main west return, sampled 600 feet west of the shaft, carried 19,200 feet of air which contained 0.08 per cent methane, or 15 cubic feet a minute, and 0.26 per cent of carbon dioxide. It will be noted that the volume of

methane on the two sides of the mine was about the same. On both sides of this mine there were extensive old workings which had been mostly sealed off. Air from one of these sealed chambers contained considerable carbon dioxide. There are faults ranging from 1 to 30 feet in displacement. The coal near some of them contained considerable gas, which passed off gradually as the coal was taken out.

MINE AT GALATIA.

The conditions in the mine at Galatia were investigated because of the isolation of the mine and its relative depth. The mine was a small one, producing only about 150 tons a day. The coal, which is the Herrin bed, is about 6 feet thick and the depth of the mine at the shaft was 370 feet. The workings were about 800 feet wide by 1,500 feet long, or about 25 acres in extent. A little gas had been observed, especially in a steep place on the third east entry on the north side. The returns were in two courses, one from the south side, and another one from the north which ventilated about two-thirds of the workings.

The south return on February 28, just south of the shaft, carried 19,100 feet of air which contained 0.08 per cent methane and 0.10 per cent carbon dioxide. Seven miners worked in this air. Little gas had ever been noted on this side, which included many old workings. The main north return near the shaft carried 13,020 feet of air which contained 0.05 per cent methane and 0.06 per cent carbon dioxide. Seventeen miners worked on this side. The volume of methane was $15\frac{1}{2}$ cubic feet per minute in the south return and $6\frac{1}{2}$ cubic feet in the north return. As may be noted from the figures given above, 50 per cent more gas came from the part of the mine in which mining was less active.

CONDITIONS IN REPRESENTATIVE MINES IN HARRISBURG BED.

MINES ABOUT HARRISBURG.

In Harrisburg and to the north and south up and down the railroad, bed No. 5 is extensively worked by two companies. At Harrisburg the depth to this coal bed is 230 feet and there is a general rise of the beds to the south and a dip to the north.

O'GARA NO. 15 MINE.

The shallowest mine examined was the O'Gara No. 15, in which the top of the coal was only 58 feet below the surface, at the bottom of the shaft. The mine produced about 1,200 tons a day and 5 feet of coal was worked. The coal is overlain by 40 feet of almost impervious shale which should be expected to confine some gas. It was stated, however, that no gas had ever been noted in this mine.

The main returns were in two airways, one ventilating the west and the other the east workings, there being about 58 miners on each side. A small part of the air of the east return passed out in a crosscut. The remainder, sampled 250 feet southeast of the shaft, had a volume of 21,460 feet and it contained 0.01 per cent methane, and 0.09 per cent carbon dioxide. The main west return, sampled 150 feet south of the shaft, had a volume of 18,360 feet and contained no methane and only 0.04 per cent carbon dioxide. That the west return should contain no methane whereas the east return contained some was probably due to the fact that the east workings extended farther to the north down the dip where the coal is under thicker cover, the conditions otherwise being the same on both sides.

O'GARA NO. 7 MINE.

At the O'Gara No. 7 mine, a mile southeast of No. 15, the coal lies slightly deeper, the shaft being $76\frac{1}{2}$ feet deep. The coal bed is 7 feet thick and a force of 90 miners produced about 800 tons of coal a day. The entire volume of return air was measured near the shaft on February 9, and found to be 20,520 cubic feet. It contained 0.03 per cent methane and 0.10 per cent carbon dioxide. These figures indicate an outflow of methane of only 6 cubic feet a minute from about 300 acres of workings.

O'GARA NO. 9 MINE.

The O'Gara No. 9 mine is a mile southwest of Harrisburg. The coal is at a depth of 156 feet and the beds dip gently to the northwest. The air of two main returns of the mine was sampled by S. O. Andros on August 18, 1912. Duplicate samples were collected in the main north entry 10 feet north of the hoisting shaft, where the volume was found to be 30,870 cubic feet per minute. It contained 0.18 and 0.14 per cent methane and 0.14 and 0.10 per cent carbon dioxide, results indicating an emanation of $55\frac{1}{2}$ and 43 cubic feet of methane a minute, respectively. The main south entry, 25 feet south of the main hoisting shaft, carried 37,800 feet of air which contained 0.03 and 0.04 per cent methane or 11 and 15 cubic feet a minute, and 0.04 and 0.12 per cent carbon dioxide. The great difference in the amounts from these two entries is a noteworthy feature. The total volume of methane in the upcast averaged 62 cubic feet a minute. At the time of sampling the mine had an average daily production of 2,518 tons.

GARA NO. 3 MINE.

The O'Gara No. 3 mine is in Harrisburg and the coal, which is 6 feet thick, lies 239 feet below the surface at the shaft. The mine is old and large and reputed to be the most gaseous one in the region; there have been many small ignitions, which have burned men and mules. The daily production was about 2,100 tons. A dike of

igneous rock crosses the mine about 500 feet southwest of the shaft and had been traced for 3,100 feet. It is 15 to 30 feet thick, stands vertical, and has converted the coal into coke for a distance of 6 to 8 feet on both sides. Another smaller dike 8 to 10 feet wide was found 2,600 feet southeast of the shaft and traced for 1,600 feet.

The air in the east return of this mine ventilated about twice the area traversed by the air in the west return. On February 10 the air of the main east return 700 feet east of the shaft was found to have a volume of 30,464 feet and it contained 0.09 per cent methane, or 27 cubic feet a minute, and 0.21 per cent of carbon dioxide. There were 132 miners in the workings ventilated by this air. Another return known as the first south entry off the east entry was sampled 150 feet due east of the shaft. It carried 4,560 feet of air, which contained 0.13 per cent methane, or 6 cubic feet a minute, and 0.10 per cent carbon dioxide. There were 90 miners in the workings on this split. A return entering by a manway 50 feet east of the shaft was found to carry 16,848 feet of air, which contained 0.12 per cent methane, or 20 cubic feet a minute, and 0.15 per cent carbon dioxide. This air came partly from the east and partly from the west and ventilated workings in which about 100 miners were employed. The main south return 200 feet southwest of the shaft was found to carry 27,936 feet of air, which contained 0.09 per cent of methane, or 25 cubic feet a minute, and 0.19 per cent carbon dioxide. The air in this return included a split of east air, which was largely from old workings and in part from about 20 fresh working places. The total volume of methane in the returns from this mine was 78 cubic feet a minute. The small amount from the first south entry off the east entry is a noteworthy feature.

O'GARA NO. 12 MINE.

The O'Gara No. 12 mine is 3 miles northeast of Harrisburg in the bottom of a basin in which the coal is slightly below sea level. The shaft depth is 421 feet and the coal worked is 5 feet thick. Gas is noticed in small amounts in places, but it has never caused much difficulty. The return air was in two main currents; one from the north workings, in which there was no mining, and the other from the south workings, where 60 miners were employed.

The main north return 150 feet north of the shaft had a volume of 17,856 cubic feet and contained 0.20 per cent methane, or 35 cubic feet a minute, and 0.15 per cent carbon dioxide. A small proportion of the south air was escaping into this part of the mine.

The main south return 150 feet south of the shaft had a volume of 18,291 feet and contained 0.02 per cent of methane, or slightly less than 4 cubic feet, and 0.05 per cent of carbon dioxide. The cause for the much greater amount of methane on the north side than on

the south side was not apparent. It was expected that the larger amount would be found in the south return where 60 miners were working.

O'GARA NO. 1 MINE.

The O'Gara No. 1 mine is a mile north of the O'Gara No. 12 mine, and, owing to a rise of the strata, the coal is only 311 feet below the surface at the shaft. The mine, which is a relatively large one, was about 210 acres in extent and had a production of 1,400 tons. About 5½ feet of coal was taken out. The return air came to the shaft in two principal airways, one from the south and the other from the north.

The main south return, sampled near the shaft on February 9, carried 18,870 cubic feet of air, which contained 0.61 per cent methane, by far the highest percentage noted in this region, and 2.07 per cent carbon dioxide, which was also exceedingly high. This air came from the southwest and east workings, where 37 miners were employed. A small part of the air escaped by other outlets. The air in the main north return from the northeast and west workings was sampled 225 feet north of the shaft. Its volume was 23,056 feet and it contained 0.36 per cent methane and 0.09 per cent carbon dioxide. About 200 men were mining coal in this air. The total volume of methane was 134 cubic feet a minute in the south return and 83 in the north return. The condition causing greater amounts of methane in this mine than in others in the vicinity was not evident unless it was the much greater area of coal exposed. The cause of the larger quantity in the south return than in the north return was not apparent.

MINES NEAR ELDORADO.

The mines in the vicinity of Eldorado work the No. 5 or Harrisburg coal, for in this region the No. 6 or Herrin bed is thought not to be sufficiently thick to be mined. At Eldorado bed No. 5 is about 400 feet below the surface or a few feet below sea level and lies in a shallow syncline with its axis extending nearly due north and south; its thickness is 4 feet 8 inches. Two mines at Eldorado were investigated, Nos. 8 and 10 of the O'Gara Co. The return air of the No. 11 mine was not sampled because it reached the upcast in many openings.

O'GARA NO. 10 MINE.

The O'Gara No. 10 mine is in the western margin of the village. The workings are extensive and 150 miners were employed. On February 10 a sample was taken of the air in the main north return near the overcast 300 feet north of the shaft. The return air had a volume of 28,944 cubic feet per minute and contained 0.35 per cent methane, or 101½ cubic feet a minute, and 0.16 per cent carbon dioxide.

The air in the main west return, which was sampled 350 feet south of the shaft, was partly from the same source as the air in the main north return, but included a return from some workings on the east side ventilated by a split of fresh air. It was estimated to carry about 80 per cent of the total ventilation from the west side of the No. 10 mine as part of the air was passing into the No. 11 mine through some holes on the east side. The volume of air in this west return was 53,460 cubic feet a minute and it contained 0.26 per cent methane, 0.139 cubic feet a minute, and 0.10 per cent carbon dioxide. From the figures it appears that the returns from this mine were discharging about 250 cubic feet of methane a minute.

O'GARA NO. 8 MINE.

The O'Gara No. 8 mine was a new and relatively small one in the southern part of Eldorado about a half mile east of the No. 10 mine. The depth was about the same, but the area of workings was only about 160 acres. A prominent igneous dike divides the mine into two parts, east and west. Along the contact of the dike and the coal considerable coke has been formed by the heat, especially on the main east gangway which crosses the dike.

The return air in the mine was tested on February 10. The main east airway off the south return at a point 300 feet due south of the shaft was found to carry 15,750 feet of air containing 0.15 per cent methane, or 24 cubic feet a minute, and 0.11 per cent carbon dioxide. Thirty-four miners were working in this side of the mine. The west airway off the south return 300 feet southwest of the shaft carried 6,930 feet of air which contained 0.26 per cent methane, or 17½ cubic feet a minute, and 0.10 per cent carbon dioxide. This current was from the west half of the mine where 70 miners were taking down coal. As possibly half of the air on that side was escaping by side holes into the west return, it was difficult to ascertain the amount of methane in separate parts of the mine. The total emanation, found by testing the upcast, was 42 cubic feet of methane a minute.

NIGGER HILL MINE.

The Nigger Hill mine at Grayson southeast of Eldorado produced about 850 tons a day and 5½ feet of coal was worked. It was regarded as a moderately gaseous mine, for it showed considerable gas in places, especially on the east side. The depth at the shaft was 336 feet. A sample of air from the main east return was sampled 300 feet southeast of the shaft on February 10. This air ventilated about two-thirds of the working area, including old workings, and 52 miners were taking down coal at the time. The volume of air passing was 38,220 cubic feet a minute, and it contained 0.23 per cent methane and 0.10 per cent carbon dioxide. The air in the main west return,

sampled 300 feet southwest of the shaft, had a volume of 28,296 feet and contained 0.10 per cent methane and 0.10 per cent carbon dioxide. The total volume of methane per minute was 88 cubic feet per minute in the east side and 28 feet in the west side, but no apparent cause for this difference could be found.

CONDITIONS IN REPRESENTATIVE MINES IN MURPHYSBORO BED.

MINES NEAR MURPHYSBORO.

In the vicinity of Murphysboro there is a group of mines that work the Murphysboro or No. 2 bed, the outcrop of which passes through the southern edge of the city. The bed dips to the north and east at a low angle, and at the Harrison mine it lies 160 feet below the surface. The mines about Murphysboro give off little methane, although traces are found in some of the workings even up to the vicinity of the outcrop. Three mines in this field were investigated, the No. 9, the Harrison, and the Blair No. 1.

NO. 9 MINE.

The No. 9 mine is on the flats of the Big Muddy River 2 miles north-east of the city. The coal here is at a depth of 108 feet. The production of the mine is about 1,400 tons, and 121 miners were employed at the time of the author's visit on March 4, 1912. The return air was sampled at two places. The first sample was taken in the main southwest return and represented the ventilation of all of the workings except those aired by a split that was sampled separately. This return from the main southwest carried 55,300 cubic feet of air, which contained 0.05 per cent methane and 0.11 carbon dioxide.

The split above mentioned, which joined the main return near the upcast was from east entries 1, 2, and 3 south, a district in which only eight miners were working. This split carried 30,800 cubic feet per minute of air containing 0.02 per cent methane and 0.05 per cent carbon dioxide. From these figures it would appear that the total outflow of methane in this mine was only 34 cubic feet a minute.

HARRISON MINE.

At the Harrison mine, which is 2 miles north of Murphysboro, the production was about 800 tons a day, and 65 miners were engaged. The mine includes a relatively large area of old workings. The total return near the fan was found to carry 65,755 cubic feet of air, which contained 0.03 per cent methane and 0.87 carbon dioxide. These figures indicate a total outflow of methane in the mine of about 20 cubic feet a minute; the great preponderance of carbon dioxide over methane in the return is worthy of note.

BLAIR NO. 1 MINE.

At the Blair No. 1 mine, which is a mile northwest of the Harrison mine, the coal is 135 feet below the surface. The production here is about 350 tons a day. There are two returns, each carrying the ventilation of about half of the mine. On March 5 the main east return carried 20,844 cubic feet of air, which contained 0.31 per cent carbon dioxide and no methane. Twenty-four miners were at work in the area ventilated by this air. The north return carried 18,120 feet of air, which contained 0.01 per cent methane and 0.33 carbon dioxide. Seventeen miners were working on the north side. These figures show that the mine was producing less than 2 cubic feet of methane per minute, whereas the carbon dioxide was more than 60 times as much.

PRESSURE OF GAS IN COAL BEDS.

In order to ascertain the pressure and in some measure also the volume of gas in the solid coal in place, tubes were inserted into holes in several mines. The mines selected for the tests were widely scattered over the area under investigation and represented considerable diversity of conditions. They were the Zeigler, United, Benton, Dering No. 11, Dering No. 18, Blackbrier, Hart-Williams, and B mines.

METHOD OF MAKING TEST.

The method used was nearly uniform in all cases. A part of the mine was selected, usually some advanced gangway heading with no adjoining workings, where the coal was most solid and presented the smallest area for escape of gas. A hole about 10 feet deep and about 3 inches in diameter was bored with an auger. Into this hole was inserted a 10 $\frac{1}{2}$ -foot length of $\frac{1}{2}$ -inch iron pipe, of which the last 3 feet was perforated. To the upper end of this pipe, which extended 6 inches beyond the face, a pressure gage was attached. The first 7 feet in the hole was tightly tamped with fire clay, which was carefully packed into place in small quantities at a time by a tamper consisting of a 2 $\frac{1}{2}$ -inch iron flange screwed to a 10-foot length of pipe of sufficient diameter to work outside of the tube to be set in place. The $\frac{1}{2}$ -inch tube carried a collar 3 feet from its inner end to hold the packing. In some holes the packing was finished with a few pounds of plaster and in nearly all holes was perfectly gas tight.

The principal features of one of these holes is shown in figure 32.

Many of the holes were gaseous while being bored, and the gas emanating would burn for several minutes. Boring and tamping were done as promptly as practicable to save as much gas and pressure as possible, and although some of the pressure was released before the valve could be set, and a certain amount of gas escaped, the losses

were not a large proportion of the total. The coal adjoining the hole was not very permeable and the pressure and volume of the gas were finally brought to normal by the flow from adjoining parts a few inches distant, as was indicated by the fact that when the pressure in the gage was released, it would again rise to the original amount in greater or less time. The volume of gas was such that the hole would fill many times after it was emptied. The area of the walls of the 3-foot chamber in a hole 3 inches in diameter is $2\frac{1}{2}$ square feet, and this factor was used in some of the calculations for the rate of gas emanation.

TESTS IN ZEIGLER MINE.

FIRST TEST

Two tests were made in the Zeigler mine—one on March 23, 1912, and another on April 7, 1912.

The tube was placed in a working face of a gangway in the extreme northeast corner of the mine, which was considerably in advance of any chamber workings. The place selected was a gaseous one, with



FIGURE 32.—Tube inserted in coal to ascertain pressure and flow of gas. The iron pipe *a* is $10\frac{1}{2}$ feet long, including a 3-foot length of perforated pipe coupled at flange *b*, above which is 7 feet of tamping. The gage *g*, screwed on at the outer end, is an ordinary steam gage reading to 50 pounds.

a hissing feeder in the top which would ignite readily and burn indefinitely. The coal was porous and wet.

When the pipe was set in place the gage recorded an initial pressure of $4\frac{1}{2}$ pounds, and $1\frac{1}{2}$ hours later the pressure had gradually risen to $10\frac{1}{2}$ pounds. Four days later it was 12 pounds, and this pressure was sustained until April 7, when the gage was removed. The volume was small, but a 2-inch flame burned for some time. On resetting the gage the pressure rose to 12 pounds, and this was repeated twice in an hour.

SECOND TEST.

In the second test at Zeigler, on April 7, the tube was placed in a hole in the solid coal in the sixth gangway west off the C entry south, in the extreme southwestern part of the mine or the opposite side from where the first test was made. The tube was in advance of all workings and in perfectly solid coal. Of course the blasting of the gangway had caused some disturbance. In boring the hole much gas was found, which would burn vigorously. After tamping the pipe in place the gage went rapidly up to 25 pounds and then more gradually to $31\frac{1}{2}$

pounds, which it recorded in 20 minutes. This pressure was sustained until April 13.

When the cock was opened, a moderate volume of gas came out and on closing it the pressure rapidly rose to 30½ pounds. During the next month the pressure increased to 33 pounds and then gradually decreased to 25 pounds, possibly because a slight leak appeared to have developed. The volume of gas at this stage was found to be small, but its flow was continuous, and it burned with a high flame.

TEST IN HART-WILLIAMS MINE.

On February 27, 1912, a tube was inserted in a hole in the solid coal in the extreme southeastern part of the Hart-Williams mine. The hole was 10 feet deep, the pipe used in this case was 1 inch in diameter, and there was 7 feet of packing, leaving 3 feet of perforated tubing in the inner end of the hole. The hole was placed horizontally in the middle of the coal bed. On the next day the gage registered about 12 ounces. On March 9 the gage was found to indicate a scarcely perceptible amount. The gage was replaced by a mercury column and the pressure was found to be 2 ounces. A sample of the gas was then drawn and on analysis it was found to contain 42.3 per cent of methane and 0.24 per cent of carbon dioxide. The gas was lighted after this sample had been drawn and it burned with an 8-inch flame for nearly an hour, when the cock was turned off. It was lighted again subsequently, but its volume was not determined.

TEST IN DERING NO. 11 MINE.

On March 28 a test was started to determine the pressure of methane in solid coal in the Dering No. 11 mine north of West Frankfort. The tube was set in a 10-foot hole in the end of the sixth southeast entry about a half mile southeast of the shaft. The pressure rose at once to 9½ pounds, and 2½ hours later it was 10 pounds. This pressure was sustained for several days. On April 13 one-half cubic foot of gas was drawn off and the pressure fell to 9 pounds; when another half foot was withdrawn, the gage reading was 7½ pounds; but when another one-half foot had been drawn, it was 8 pounds. Then after 10 minutes the pressure rose to 9½ pounds and finally to 10 pounds where it remained stationary until the gage was taken off a few hours later. A sample of this gas had the following composition:

Composition of gas from test hole in Dering No. 11 mine.

	Per cent.
Methane	31. 19
Carbon dioxide	. 29
Oxygen	16. 51
Nitrogen (by difference).....	52. 01
	100. 00

TEST IN DERING NO. 18 MINE.

On April 4, 1912, a test was started of gas pressure in solid coal at the Dering No. 18 mine northeast of West Frankfort. The place selected for the test was near a strong feeder. A 9½-foot hole was bored in the face of the first southwest entry where it yielded a strong flow of gas. When the tube had been set and the gage applied the pressure at once rose to 15 pounds, in 5 minutes to 20 pounds, and in another 5 minutes to 24½ pounds, where it remained stationary. On April 5 the gage reading was 23½ pounds; on April 6, 24½ pounds; and on April 7, 24½ pounds, where it remained until the time of the author's return on April 11. When the cock was opened the pressure dropped to zero in about 40 seconds, but when the cock was closed, the pressure rose to 23 pounds in 15 minutes. A sample of the gas taken at this time was found to contain only 0.32 per cent of methane and 0.24 per cent of carbon dioxide. The drawing of the sample reduced the pressure to zero again but in a few minutes it rose to 6 pounds, where it remained a few hours until the gage was removed. The gas volume was small, for the removal of 1 cubic foot caused the pressure to drop to zero.

TEST IN UNITED MINE.

On March 22, 1912, a test was made of the pressure of gas in solid coal in the United mine east of Christopher. The hole was bored in the end of the sixth gangway north off the main entry east or in the extreme northeast corner of the mine. When the tube had been set the gas flowing from it could be ignited but when the gage was attached no pressure was developed. Observations at intervals to March 30 showed no pressure and finally, when the mercury gage was applied, the pressure was not perceptible. It was also impossible to collect any appreciable volume of the gas in a bottle arranged for measuring volume. However, when the gage was opened the gas burned with a flame 4 inches long; seemingly it would have continued burning for some hours and possibly for a much longer time.

TESTS IN BENTON MINE.

Early in April a test was made of the gas pressure in the coal at the Benton mine. A hole 9 feet deep was bored in the end of the seventh gangway west off the main south entry. The coal was very dry and solid and there was a slight hissing of gas in the face. No pressure was developed in a half hour, and on the author's return a week later there was still no pressure showing in the steam gage, nor did a mercury gage show any. No gas came from the cock when it was opened.

Another tube was set in the coal on May 4 by the mine superintendent. It was set in a hole $9\frac{1}{2}$ feet deep, $6\frac{1}{2}$ feet being well tamped. The hole was in the fifth gangway south off the seventh west entry south. The pressure was 16 pounds, with a small leakage around the tube. After this leak had been closed the pressure rose to $16\frac{1}{2}$ pounds, which was sustained until May 13 when the gage was removed.

TESTS IN CHICAGO & CARTERVILLE CO. "B" MINE.

Two tests were made of gas pressure in the solid coal in the Chicago & Carterville Co. "B" mine, 3 miles northeast of Herrin.

The first test was made in the fourth heading west off the first gangway north off the main east entry in the end of a working with no chambers in the vicinity. Gas was hissing from the face at the place selected, which was regarded as very gaseous. The hole was like the others, 10 feet deep with 3 feet of perforated pipe at the end and 7 feet of tamping. The gas burned with a $1\frac{1}{2}$ -inch flame from the end of the pipe before the gage was set. The gage did not move at first but in a few hours pressure was manifested and the next day was 7 pounds. The pressure was sustained at 7 pounds the first week and then increased to $7\frac{1}{2}$ pounds, where it remained until June 3 when the gage was removed.

Another gage was set in the end of the second gangway south off the west entry. The coal was wet but solid. No pressure was developed at first but after 24 hours the gage showed $2\frac{1}{2}$ pounds and this was sustained until June 3 when the gage was removed.

TEST IN BLACKBRIER MINE.

On April 3, 1912, a tube was set in the solid coal at the end of the most advanced gangway in the Blackbrier mine. The direction of the hole was west, parallel with the principal jointing. The hole showed no gas and the gage gave no indication of pressure. A week later it was found that there was still no pressure, even to a mercury gage, and no gas was released when the tube was opened.

COMPARISON OF RESULTS IN DIFFERENT MINES.

A tabulation of the initial pressures recorded soon after closing the gage, and the pressures developed at later periods, is given in the table following.

Comparison of pressures of gas in solid coal in 11 mines.

Mine.	Place where test was made.	Initial pressure.	Later pressure.	Time between readings.	Remarks.
		<i>Pounds.</i>	<i>Pounds.</i>	<i>Days.</i>	
Hart-Williams.....	Southeastern part.....	$\frac{1}{2}$	$\frac{1}{2}$	10	42.3 per cent methane, moderate volume.
Dering No. 11.....	Southeast entry.....	9 $\frac{1}{2}$	10	6	31.2 per cent methane, moderate volume.
Dering No. 18.....	Southwest entry.....	24 $\frac{1}{2}$	24 $\frac{1}{2}$	7	Air with 0.32 per cent methane, small volume.
United.....	Northeast corner.....	0	0		Gas would burn.
Zeigler.....	do.....	10 $\frac{1}{2}$	12	4	Volume small.
Do.....	Southwest corner.....	25	33	20	Volume very small.
Benton.....	Seventh gangway west off main entry south.	0	0		No gas.
Do.....	Fifth gangway south off seventh entry southwest.	16	16 $\frac{1}{2}$	9	Considerable gas.
Chicago & Carterville Co. "B" mine.	Northeast section.....	7	7 $\frac{1}{2}$	8	Do.
Do.....	Southwest section.....	0	2 $\frac{1}{2}$	1	Small volume.
Blackbrier.....	End of west gangway..	0	0		No gas.

VARIATIONS IN VOLUME OF GAS DURING MINING.

For the purpose of ascertaining the variations in the amount of methane at different times during periods of active mining, a second series of samples was taken in five of the mines previously visited, in which mining had been steadily carried on in the meantime. They were all mines that gave off considerable amounts of gas and were large producers.

The results obtained from the two sets of samples are compared in the following table:

Differences in amounts of methane from working mines at different times.

Mine.	Date of sampling.	Volume of methane per minute.							Daily production.
		Total upcast.	North return.	West return.	South return.	East return.	North-west return.	North-east return.	
	1912.	<i>Cu. ft.</i>	<i>Cu. ft.</i>	<i>Cu. ft.</i>	<i>Cu. ft.</i>	<i>Cu. ft.</i>	<i>Cu. ft.</i>	<i>Cu. ft.</i>	<i>Tons.</i>
Hart-Williams.....	Jan. 15	90				64			2,400
Do.....	Feb. 14	133				65			2,400
Benton.....	Jan. 16	202	95	44	53				2,100
Do.....	Mar. 9	194	68 $\frac{1}{2}$	74	52				2,100
Zeigler.....	Feb. 2	153	93		60				2,300
Do.....	Mar. 23	181	154		27				2,300
Dering No. 11.....	Jan. 19	^a 204		83		82	33	21	1,500
Do.....	Mar. 28	^a 271		91		67	32	24	1,500
Wilmington-Star.....	Jan. 19	171	86		86				1,500
Do.....	Mar. 28	212	119		93				1,500

^a Shaft sample; the other totals are sums of separate returns. The cause of the discrepancy in the Dering No. 11 mine on Mar. 28 is not known.

These results show marked variations in the methane emanation not only in the total upcasts but in the separate returns, which were undoubtedly caused by variations in the amount of gas in the coal as mining progressed, increasing in some districts and diminishing in others without relation to the tonnage removed which continued nearly uniform.

EFFECT OF CESSATION OF MINING ON VOLUME OF METHANE.

All the mines were closed by a strike on April 12, 1912, and mining was discontinued for a month. Advantage was taken of this opportunity to ascertain the effect of no mining on the amount of methane by resampling the returns. Certain mines visited presented considerable variety as to size, depth of cover, and location, all in the Herrin coal bed. The observations made are given in the following table, together with the amount of methane found in previous visits:

Volumes of methane given off while mines were shut down, compared with volumes during operation.

Mine.	Return.	Volume of methane a minute.		Daily production of mine.	Depth at shaft.
		During operation of mine.	After mine suspension 5 to 15 days.		
		<i>Cu. ft.</i>		<i>Tons.</i>	<i>Fect.</i>
Benton.....	Total upcast.....	202-194	164	2,400	630
Do.....	Main north.....	95-68½	40		
Do.....	Main west.....	44-74	82		
Do.....	Main south.....	53-52	42		
Zeigler.....	Total upcast.....	153-181	78		418
Do.....	South.....	60-27	22	2,300	
Do.....	North.....	93-154	56		
Paradise.....	Total return.....	101	41		380
Dering No. 11.....	Total upcast.....	204-271	177	1,500	500
Do.....	Main west.....	83-91	57		
Do.....	First northwest.....	83-32	27½		
Do.....	Main east.....	82-67	68		
Do.....	Third northeast.....	21-24	20½		
Dering No. 18.....	Total returns.....	49	37½		512
Do.....	Main south.....	11	15	500	
Do.....	West off south.....	19	10		
Do.....	Main north.....	19	12½		
Wilmington-Star.....	Total returns.....	172-212	130		440
Do.....	South.....	85½-93	69	1,200	
Do.....	North.....	86½-119	61		
Standard.....	Total return.....	56	62		276
Do.....	East.....	37	32	1,000	
Do.....	West.....	19	30		
Blackbrier.....	Total return.....	55½	31	1,100	185
Big Muddy.....	Total return.....	14½	3½	1,500	100
Do.....	East.....	11½	3		
Do.....	West.....	3	½		
Hemlock.....	Total returns.....	9½	15		155
Do.....	East.....	2½	8	700	
Do.....	West.....	7	7		
Taylor No 2.....	Total returns.....	12	0	1,500	140
Do.....	West.....	6½	0		
Do.....	East.....	5½	0		
Chicago & Carterville Co. "A" mine.....	Total returns.....	227½	128½	2,100	186
Do.....	Main north.....	83	21		
Do.....	Third west south.....	8	14		
Do.....	First east north.....	33	52		
Do.....	Second west south.....	35	27		
Do.....	Main south.....	68½	14½		
Chicago & Carterville Co. "B" mine.....	Total returns.....	101	140	1,500	247
Do.....	Main east.....	47	42		
Do.....	Main west.....	54	98		
Big Muddy Coal & Iron Co. No. 7.....	Total returns.....	67½	37	2,000	150
Do.....	Southeast ½.....	6	3		
Do.....	Southwest ½.....	3	½		
Do.....	Northeast ½.....	34	18		
Do.....	Northwest ½.....	24½	15½		

• Not being mined at time of sampling.

Nearly all these mines showed a marked diminution of volume of methane after mining had been suspended. In the Taylor No. 2 mine the amount dwindled to zero, in the Big Muddy mine from $14\frac{1}{2}$ to $3\frac{1}{2}$ cubic feet a minute, in the Zeigler mine from 181 to 78, in the Wilmington-Star mine from 212 to 132, and in the "A" mine from 227 or more to $128\frac{1}{2}$. In the Chicago & Carterville "B", Standard, and Hemlock mines the volume increased for some unknown reason. The total emanation in the mines given in this list was 1,551 cubic feet of methane a minute at the times of sampling in January, February, and March, whereas in April, after the shutdown, the total volume had diminished to 1,048 cubic feet, or about 32 per cent less.

RATE OF ESCAPE OF GAS FROM ILLINOIS COAL.

A few tests have been made to ascertain the amount of gas contained in samples of coal from southern Illinois with results very similar to those obtained from samples from other regions. Porter and Ovitz ^a made some extended tests of the rate of escape of gas from samples of coal from the O'Gara No. 9 mine near Harrisburg and from the Hart-Williams mine, at Benton, Ill., and also observed the capacity of the samples for oxygen absorption.

The mine samples weighed about 32 pounds and represented the clean, fresh working faces from top to bottom. The material was first crushed to pass through a $\frac{1}{2}$ -inch screen and then placed in a 5-gallon bottle which was sealed as quickly as possible, the whole operation taking about an hour. During the tests the coals were kept in the laboratory of the Bureau of Mines at Pittsburgh at temperatures varying from 54° to 90° F. The sample of Harrisburg coal was connected with an air supply by means of an air-inlet tube which extended from a reservoir of air to the center of the bottle through a hole in the cork. The bottle of Benton coal was connected with an oxygen supply admitted at the top and the gas was drawn off at the bottom into a reservoir. The quantity of air or oxygen admitted was measured and the gas was measured and analyzed. The results are given on the following page.

^a Porter, H. C., and Ovitz, F. K., The escape of gas from coal: Technical Paper 2, Bureau of Mines, 1911, 14 pp., 1 fig.

Gases given off and oxygen absorbed by two Illinois coals.

Place where coal was obtained.	Weight of sample.	Period of time.	Gas given off per cubic foot of coal.		Oxygen admitted per cubic foot of coal. ^a	Oxygen absorbed per cubic foot of coal.
			Methane.	Carbon dioxide.		
O'Gara No. 9 mine, near Harrisburg.	Pounds. 36	First 4 days.....	Cu. ft. 0.218	Cu. ft. 0.001	Cu. ft. 0.148	Cu. ft. 0.145
		Next 24 days.....	.340	.001	.148	.145
		Next 31 days.....	.217	.003	.650	.610
		Next 30 days.....	.125	.000	.244	.235
		Next 153 days.....	.372	.030	1.553	1.520
		Next 31 days.....	.029	.019	.360	.343
		9 months.....	1.083	.054	2.955	2.853
Hart-Williams mine, near Benton.	27	First 15 days.....	0.763	0.010	0.217	0.220
		Next 15 days.....	.113	.000	.490	.494
		Next 62 days.....	.577	.019	3.587	3.532
		Next 30 days.....	.107	.018	1.169	1.113
		Next 31 days.....	.065	.007	1.652	1.576
		Next 365 days.....	.196	.079		
		17 months.....	1.821	.133	7.115	6.935

^a Oxygen from air in Harrisburg coal and 95 per cent pure in Benton coal, so that results are not comparable.

^b Total for 6 months.

The irregularities in oxygen absorption were due to the irregular rate of admission. Doubtless much more would have been absorbed if the admission had been regular.

In an investigation of occluded gases in Illinois coals, Barker ^a tested a number of samples of coal from the area discussed in this bulletin. They had been partly air-dried and were placed in jars with much air. One sample from the Davis mine near Duquoin and another from the Big Muddy mine near Marion gave off no methane, one from the O'Gara No. 8 mine near Eldorado gave off 0.38 volume, and one from the Squirrel Ridge mine near Herrin gave off 0.17 volume.

Samples of drillings were collected at intervals from a horizontal drill hole bored for making pressure tests in the Hart-Williams mine. The first sample was taken at a depth of 3 feet, the second at 6 feet, and the third at 10 feet. They were sealed in tin cans. A month later they were tested in the laboratory of the Bureau of Mines in Pittsburgh, with the results following.

^a Barker, Perry, The occluded gases in Illinois coals: Ill. Geol. Survey Bull. 14, 1908, pp. 204-210; Trans. Am. Inst. Min. Eng., vol. 40, pp. 24-31, 1909.

Results of analyses of gas given off by coal samples from bore hole in Hart-Williams mine, Benton, Ill.

(G. A. Burrell, analyst.)

Sample.	Gas given off.			
	Methane.	Carbon dioxide.	Oxygen.	Nitrogen.
	Per cent.	Per cent.	Per cent.	Per cent.
1.	21.35	Less than 0.1	2.63	77.02
2.	5.80	do.	2.53	91.67
3.	10.60	do.	2.18	87.21

These results throw no light on the volumes given off; moreover, considerable air was taken in when the samples were placed in the can. The figures, however, show that methane was evolved, probably in different amounts, from the three samples and that some of the oxygen had been absorbed.

RELATION OF METHANE EMANATION TO DEPTH.

It is well established that in most areas where the coal is near the surface little gas remains in it, and in general up to a moderate depth the deep mines are more gaseous than shallow ones. The escape of gas from the coal to the surface has been in progress for an extremely long time and even from deep-seated beds there has been some loss in this way. An important factor affecting the rate of escape is the relative permeability of the strata overlying the coal, and the degree of permeability depends not only on the texture of the rock, but also on the extent to which faulting and joint crevices are present. A bed of gaseous coal near the surface overlain by sandstone would be likely to lose much of its gas, whereas a cover of shale or clay would greatly impede the escape of gas.

Interstitial moisture also impedes the movement of gas through the strata, but the extent of its influence is not well known. Fracturing of the strata is also an important factor, for the gas travels along fault planes and joint cracks. A coal bed lying several hundred feet below the surface is usually overlain by alternations of the clays and sandstones that compose the coal measures. As much of this material is only slightly permeable, there is with increasing depth a proportionate general increase in the amount of gas confined in the coal; but, as shown by figure 33, the relation is not a close one and this increase ceases below a certain depth. The table on page 224 shows the relative volumes of methane given off by mines of southern Illinois, arranged in order of their depth, and also the amount in relation to daily coal production from each mine.

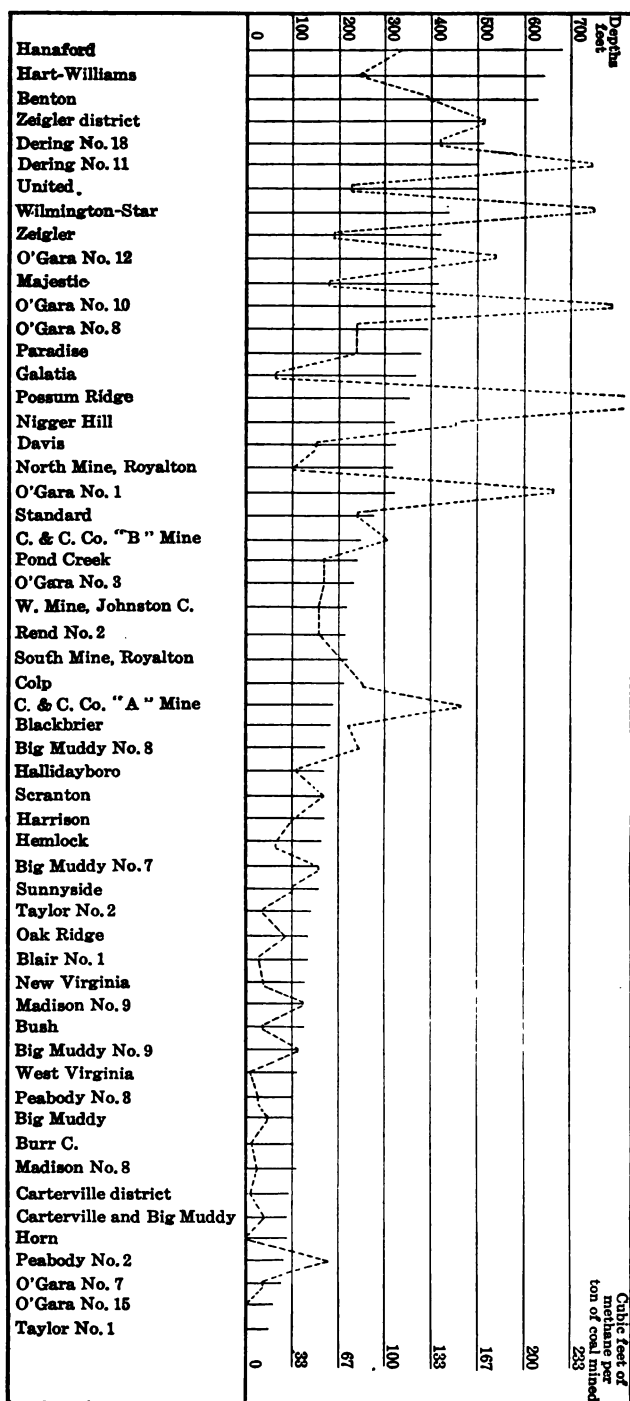


FIGURE 33.—Relation between depth of mines and amount of methane per ton of coal mined in southern Illinois.

Total methane emanation from southern Illinois mines arranged in order of their depth.

Mine.	Depth at shaft.	Methane.		Daily production of coal.
		Volume per minute.	Volume per ton of coal mined.	
	<i>Feet.</i>	<i>Cu. ft.</i>	<i>Cu. ft.</i>	<i>Tons.</i>
Hannaford.....	606	75½	109	1,000
Hart-Williams.....	640	90-133	53-80	2,400
Benton.....	630	194-202	133-138	2,100
Zeigler district.....	517	241	173	2,000
Dering No. 18.....	512	49	141	500
Dering No. 11.....	500	204-271	195-260	1,500
United.....	500	220	74	4,300
Wilmington-Star.....	440	172-212	205-255	1,200
O'Gara No. 12 a.....	421	39	186	300
Zeigler.....	418	153-181	95-113	2,300
Majestic.....	412	165	110	2,150
O'Gara No. 10 a.....	411	250	313	1,150
O'Gara No. 8 a.....	407	42	81	750
Paradise.....	390	101	81	1,800
Galatia.....	365	22	21	150
Possum Ridge.....	357	301½	563	1,000
Nigger Hill a.....	325	116	196	850
Davis.....	320	96	96	1,400
North mine, Royaltan.....	315	36	35	1,600
O'Gara No. 1 a.....	311	217	222	1,400
Standard.....	278	56	51	1,000
Chicago & Cartersville Co. "B" mine.....	247	101	97	1,500
Pond Creek.....	240	36	53	1,000
O'Gara No. 3 a.....	233	78	54	2,100
West mine, Johnston City.....	220	105	53	2,800
Rend No. 2.....	215	73	53	2,000
South mine, Royaltan.....	212	12½	72	250
Colp.....	210	24	86	400
Chicago & Cartersville Co. "A" mine.....	186	227-4	156	2,100
Blackbrier.....	185	55½	73	1,100
Big Muddy No. 8.....	167	139	80	2,500
Hallidayboro.....	160	29	35	1,200
Scranton.....	160	25	55	650
Harrison b.....	160	26	36	800
O'Gara No. 9 a.....	158	62	36	2,515
Hemlock.....	155	9½	20	700
Big Muddy No. 7.....	150	67½	49	2,000
Sunnyside.....	150	42½	31	2,000
Taylor No. 2.....	140	12	11	1,500
Blair No. 1 b.....	135	2	8	350
Oak Ridge.....	135	18½	25	1,100
New Virginia.....	120	16½	21	1,100
Madison No. 9.....	120	60	40	2,500
Bush.....	118	11½	11	1,500
West Virginia.....	108	2½	3	1,200
Big Muddy No. 9 b.....	108	34	35	1,400
Peabody No. 3.....	100	14	8	2,400
Big Muddy.....	100	14½	14	1,500
Burr C.....	100	4½	4	1,500
Madison No. 8.....	100	9½	8	1,500
Cartersville district.....	96	3	3	1,450
Cartersville & Big Muddy.....	86	8	11	1,000
Horn.....	80	0	0	
Peabody No. 2.....	80	55-4	66	1,200
O'Gara No. 7 a.....	76½	6	11	800
O'Gara No. 15 a.....	58	2	3	1,200
Taylor No. 1.....	55	1½	2	1,000

a Harrisburg bed.

b Murphysboro bed.

PROBABLE INCREASE OF GAS IN DEEPER OR LARGER MINES.

A question of practical importance considered in the investigations in Illinois concerns the amount of gas to be expected in the deeper parts of the field, as yet undeveloped. The danger of gas and the added expense that must be incurred in combating large volumes of it are important considerations affecting the development of the

coal in many areas. The results of this investigation show that the deeper mines are not likely to find more gas than has been encountered in those of moderate depth. It is found that below 150 or 200 feet beneath the surface depth is not the principal factor affecting the volume of gas, but that variations in the character of the coal cause differences in the amount given off. This regional variability also indicates that a mine with low gas volume must not be conducted with a false sense of security, for a gaseous district may be entered at any time and the gas become dangerous on very short notice. Accordingly equipment should be provided capable of coping with sudden developments of greatly increased volumes of gas.

Another important consideration is whether or not there will be an important increase in gas volume as the workings become larger and the area of exposed coal surface consequently increases. Several of the companies have already extended their mining operations over nearly all of the area owned by them, and the increase of total gas volume appears not to have increased proportionately to the area worked. This result is due to the fact that the larger part of the gas has been given off from freshly exposed coal in the working faces during the mining so that the old pillars do not contribute any large proportion of the total methane emanation. As mining progresses in the larger properties the disposition will be not to extend the workings to great distances, but to move to the adjoining area and sink a new shaft, as the item of long underground haulage is an important one in the cost of production.

SUMMARY OF CONDITIONS OF OCCURRENCE OF METHANE.

Although many observations have been made as to the conditions under which explosive gases occur in coal and are given off by it, some of the causes of the variations in the character, amount, and pressure of such gases are still unknown. In this chapter are summarized the principal facts developed by the investigations in Europe and this country, including the author's own observations in Pennsylvania and Illinois.

COMPOSITION OF GAS GIVEN OFF BY COAL.

The gas given off by coal in mining as well as in laboratory experiments under different conditions shows considerable range in composition. Although methane is the principal component its proportion varies not only in gases in the mine but in gases extracted from coal samples under various conditions. In most cases the gas given off by coal and by blowers contains more than 80 per cent of methane, and in places the methane is nearly pure. Carbon diox-

ide, nitrogen, and oxygen, with proportions varying greatly, are the other principal ingredients of the gas from fresh coal. The variations in composition doubtless indicate a complex chemical process in the development of the coal, which is further indicated by the great variation of volume of contained gases in the coal from bed to bed and from place to place in the same bed. These variations have not as yet been connected with visible characteristics in the coal, and they appear to be entirely independent of the chemical constitution of the coal, reported as "fixed carbon," "volatile combustible matter," etc.

The gas given off by coal at different stages of exposure and in fragments of various sizes also shows great variability. In the mine there is considerable difference between the composition of gas from blowers and from the fresh face and that from old pillars and ribs, and in the laboratory the degree of pulverization and the duration of tests materially affect the composition of the gases obtained. Doubtless air absorption with resulting oxygen combination has much to do with this phase of the problem.

MANNER OF ESCAPE OF GAS FROM COAL.

Seemingly gas escapes from coal by percolating through the pores or interstices between the coal grains and also through fissures great and small. The principal controlling factors under these conditions therefore are, (1) texture or size of grain, (2) extent and size of fissures, and (3) amount of pressure forcing the gas outward. Gas escapes most rapidly when the mining operation reaches a fissure or a body of shattered coal that constitutes a reservoir under high pressure and when such a body of coal is thrown out in such large quantity that much of the gas is given off at once. Outbursts of this character have caused numerous accidents, some with considerable loss of life. Very careful study of these outbursts has not revealed any definite law of occurrence and no dependable means has been found for detecting their proximity or for preventing them. They happen most often in zones where the strata are crushed or disturbed or in places where the coal thickens and thins, but they are not connected with any peculiarities of the coal that are discernible according to our present knowledge. Bodies of softer, more porous coal inclosed by harder coal are naturally very likely to be a source of outbursts, for they may contain a large volume of gas dammed back by the hard coal. The likelihood of increased gas emanation and accumulation from squeezes and other local movements in mines is well recognized, for ordinarily these accidents not only cause deep fissuring but interfere with ventilating currents.

PRESSURE OF GAS IN SOLID COAL.

The pressure of gas in the solid coal has been determined in England, France, Belgium, Austria, and Illinois with results presenting anomalous variations. The tests by Wood in England showed pressures ranging from 176 to 461 pounds in the deeper test holes and, though in several tests the pressures were related to the depth of hole, there was great variation between closely grouped localities in each mine and between two beds in different mines. In Austrian mines the variation was from 82 to 142½ pounds. The test in two inclined beds in the Belle-Vue mine in Belgium, made in a heading 2,264 feet below the surface in holes that reached the coal from a heading in rock, showed pressures of 325 to 637 pounds. Tests made when this heading was extended into the coal showed little immediate fall of pressure, in most cases proving that pressure was sustained for a considerable time by the very imperfect permeability of the coal. This condition also accounts for the slowness with which the higher pressures are manifested, because the adjustment of equilibrium of pressure takes place slowly where the permeability is slight. Several observers have noted that the highest pressures are not found in places where the most gas is noticeable in the workings, and this was the case in headings where the author made pressure determinations in southern Illinois. In nearly all pressure tests it was found that when the cock was opened so that pressure was lost, it was quickly regained when the cock was closed. Further evidence of exceedingly slow permeability of the solid coal to gas was afforded by the tests in the Beaulieusart mine in Belgium. In that mine, with pressures of 120 and 187 pounds in two holes, several gas and coal outbursts only a few yards distant caused only slight oscillations in the pressure in one hole, although they finally drew off the pressure in the nearer hole. In the test holes made by Simon and Morin in the Liévin mine, France, which showed pressures of 10 to 105 pounds, the places giving off the most gas showed low or moderate pressures.

The observations reported by Ghysen showed that pressure was not closely related to obvious gas emanation for in only slightly gaseous workings at Grand Hornu and Marihay the pressures were 60 and 225 pounds, and in a very gaseous bed in the Agrappe mine the highest pressure observed was less than 15 pounds.

In the many test holes in St. Étienne it was found that in general the pressure and gas volume increased regularly with depth of holes, but the highest pressure was only 44 pounds, even in holes 23 feet deep.

Holes to ascertain the possibility of draining the gas from solid coal proved their entire uselessness on account of the small surface of

coal exposed. The pressure of water in the strata overlying coal beds is probably a factor of some importance in gas pressure and also the weight of the strata which is very great in a mine 2,000 feet deep.

The idea of Wood and Mallard that there is a precise ratio between pressure and depth of hole is not sustained by most observations. The local variations in pressure show that the gas is irregularly distributed in coal beds even in short distances, but of course in general the pressure gradually increases as the solid coal is penetrated beyond the influence of fissures and ordinary outflow to the surface, and it attains a maximum some distance from the face. All the investigations have shown that this maximum also varies greatly in the same bed from place to place. Pressures observed in the Illinois coal field ranged from a few ounces to 33 pounds and in four headings in solid coal no pressure at all was indicated. Pressure from a blower in a Pennsylvania anthracite mine was found to be 45 pounds to the square inch.

The pressure tests indicated that in general high pressure, if present at all, is manifested where the coal is most compact, and owing to leakage near the face the higher pressures are found only in the deeper test holes, the depth necessary for a high pressure depending on compactness of the coal.

VOLUME OF GAS IN COAL.

The volume of gas in coal as shown by the amounts given off by mining a definite amount of coal and by laboratory tests of coal samples is as variable as the pressure. Variation in the original volume is no doubt related to the organic character and history of the coal and to other factors not yet ascertained. In many cases in the mines the coal containing the most gas, or gas with highest pressure, does not have the most gaseous workings, or gas bodies dangerous to the miners, for a large volume may be given off so regularly and for such a long time that its outflow is less noticeable than the rapid emission of a relatively much smaller amount. Moreover, we can not know the original volume of gas in different coals, for varying amounts have been lost by escape through overlying strata as well as in opening and mining. Some of the tests have shown that in many instances less total gas a day is given off in workings showing marked accumulations of gas in the faces, or from feeders, than in workings in which the outflow is general but so gradual that its presence is not suspected when the ventilation is adequate.

The conditions under which large volumes of gas are contained in the coal, such volume as 1 to 1 and 3 to 1 or more, have not been determined. Possibly the gas is locked in some loose chemical combination, dissolved as salt dissolves in water, or condensed as

in porous charcoal, but undoubtedly it fills the pores and crevices in which it is condensed considerably by pressures of 50 to 500 pounds to the square inch. The variation in volume of methane emanation in mines has been studied in various mines in Europe and was closely considered in the author's series of observations in the mines in Pennsylvania and Illinois. If only the deeper workings where gas has had little chance to escape to the surface be considered, there are found great differences in the amount of gas given off by mining in different beds and different workings in the same bed. As the investigations progressed it was soon found that these differences were connected neither with variations in structure of the measures nor with cover below the first few hundred feet, but were local features in the coal. One excellent instance of this was in the 5-foot workings at the Lance mine, Plymouth, Pa., where the chambers on the east side were giving off 400 cubic feet of methane a minute and those on the west side were yielding 70 cubic feet, with area of coal exposed and other conditions closely similar on both sides, except that on the less gaseous side nearly twice as much coal was being mined. A similar difference but in less degree was found in the Hillman workings next above.

Most of the published facts relating to mine gas give proportions of methane in the return only; others indicate the volume from the mine or from different returns, and a few observers calculate the volume to the ton of coal mined or to the area of coal exposed. Some of the investigations show the differences in amount of methane given off under different conditions of mining, periods of cessation from mining, and changes of atmospheric pressure. Many figures of volume of methane carried by air returns in a given time demonstrate regional variations that indicate that some districts or some mines are especially gaseous and that generally in a mine certain beds or certain workings in a bed give off much more gas than others. In places there is close ratio between gas volume and amount of coal mined or area of coal surface exposed. This latter relation is exceptional, however, and the volume of gas varies from place to place without connection with any special properties of the coal known to us at present.

OUTFLOW AT CERTAIN MINES.

Among mines with large outflow of gas those near Wilkes-Barre, Pa., lead with 2,000 to 3,500 cubic feet a minute, and some of the more fiery of the Prussian mines range from 400 to 625 cubic feet. Figures of this character have to be considered in connection with extent of workings and tonnage of output and the data for these factors are given in but few instances. In the Saarbrücken basin the volume varied from 16½ to 2,160 cubic feet to the ton of coal

mined; in the gaseous mines near Wilkes-Barre the rate ordinarily ranged from 2,000 to 3,850, with an average of about 1,500 cubic feet a ton; in some Austrian mines it ranged from 519 to 7,469 cubic feet; at Gemeinschaft and Ath-Gouley the rate per ton was 409 to 3,360 cubic feet, and in two fiery French mines the figures given are 883 and 1,377 cubic feet. In the deeper mines in southern Illinois the amounts varied from 50 to 260 cubic feet per ton. In some mines, however, the amount did not diminish very materially when mining was stopped for a time, especially in certain workings in mines near Wilkes-Barre and in Austria, but in others the diminution was noticeable, even the Sunday cessation making an appreciable difference. Fresh face workings yielded vastly more methane than workings in pillars. The amount of gas per 1,000 square feet of coal surface determined at a few mines near Wilkes-Barre and Scranton ranged from 0.23 to 5 cubic feet a minute, with amounts of 1.67 to 5 cubic feet from workings that were especially gaseous. In the mines at Liévin the range was from 2.3 to 14.7 cubic feet. The coal in test holes at St. Étienne gave off gas varying in volume from 20 to 88 cubic feet a minute, calculated for an area of 1,000 square feet, the amount in general increasing as the pressure increased. In test holes made by Wood, the maximum volumes calculated for 1,000 feet of area ranged from 4 to 100 cubic feet a minute without relation to pressure.

INFLUENCE OF TEMPERATURE ON METHANE EMANATION.

Many tests have been made to ascertain the amount of methane given off by coals at ordinary temperature and at 212° F., in a vacuum, in the presence of various other gases, and in different periods of time. The tests have yielded a wide range of results without revealing any special relations. The volumes of gas and the percentages of methane differ greatly with different coals and in some mines from nearby samples of the same bed. Ordinarily, with increased time, there is increased volume as tests of six months and longer have shown, and volume or celerity of emanation increases in proportion to the fineness to which the sample is ground. The relative volume of gas to coal ranged from 0.1 to 8 times at 212° F., and to 5.5 times at ordinary temperature, but the average amount probably falls below 1 under ordinary temperature. A sample of anthracite from Pennsylvania gave off nearly four volumes in a long test. Nearly all kinds of coals, according to our present ideas of classification, showed great range in gas volume. Samples of certain coals from especially gaseous chambers in mines gave off only moderate volumes of gas when tested in the laboratory, but no determinations have been made as to what extent coals testing high in gas cause more gaseous workings than the others.

INFLUENCE OF COMPOSITION OF COAL ON GAS EMANATION.

Doubtless the local differences in gas content were related to variations in putrefactive and other conditions under which the coal was formed, such as variations in character of the vegetable deposits and the nature and time of deposition of the overlying strata. Many of the analyses given by European writers show no relation between methane and composition of the coal, as the latter is usually reported, and we know that lignite, anthracite, and bituminous coal are closely alike as to their methane component and its variation from place to place.

Possibly when we can ascertain the organic components of coal, and when gas distribution is studied in that connection, we shall find certain relations, especially if knowledge is available as to the original organic chemical history of coal development.

The origin of methane in coal has been considered by many writers and although it is evident that the gas is a product of the decomposition or molecular rearrangement of the original organic matter, the precise nature of the chemical process has not been ascertained. It is possible that somewhat similar amounts of gas were given off in the formation of all coal and the retention of the methane has depended on conditions of deposition and cover. On the other hand, it is likely that different plant components may have developed very different amounts of gas in their conversion into coal, and temperature, moisture, and rate of accumulation may have been factors. If a certain coal contains many times its volume of gas in voids that can not be more than 10 per cent of the volume of the mass there must be great condensation under high pressure or some peculiar molecular relation that we do not understand. Tests of powdered coal prove that a long time is required for all the gas to leave the coal, however great the quantity liberated at first exposure.

ATMOSPHERIC INFLUENCES.

In this report there can be presented only a part of the extensive literature on the influences of barometric changes on gas emanation. Although some of the investigations appear to show that such relation has no practical importance, others indicate that a largely increased volume of methane results from marked diminution of atmospheric pressure, especially in returns draining old workings. The argument frequently advanced that atmospheric pressure can not affect the emanation of gas from the solid coal because of the high pressures as shown in test holes is entirely fallacious for whatever the pressure may be 10 feet in the coal it must become zero at the face where the gas actually escapes. The curves drawn by Galloway, Chesneau, Koehler, Schöndorf, Nasse, Hilt, Broockman, and Morin undoubtedly indicate a close relation between methane emanation and atmospheric

pressure in the mines that were tested, and naturally it is to be expected that this relation will be found in other mines of the same character. However, it appears likely, as maintained by most of the French and English engineers, that the increased emanation at the time of low barometer can not be a source of danger in well-ventilated mines. Without suitable ventilation carried to gaseous working faces there is danger of explosions at any time, and if the danger is slightly greater with diminished atmospheric conditions it is merely one of degree. There are in most gaseous districts many chambers that rapidly fill with gas if the air current fails owing to open doors or lack of suitable bratticing.

RELATION OF METHANE ACCUMULATIONS TO EXPLOSIONS.

Deductions based on statistics of explosions in mines as related to the weather or other factors are misleading for many such explosions have not been caused by increased methane emanation. In the first place we now know that many of the larger explosions were propagated mainly by coal dust, and in the second place most explosions are started by accidental ignition, generally without connection with any unusual gas outflow in the workings as a whole. Fortunately many gas outbursts and local gas accumulations are not ignited because of increased precautions. The accumulation of local bodies of explosive gas is of every-day occurrence in parts of many mines and in those places carelessness of an individual may cause an accident without any connection whatever with atmospheric pressure, temperature, humidity, earth tremors, or the general volume of methane in the mine. Many of the most dangerous gas bodies are detected by the fire boss on his early morning tour and provision is made for their removal. It has not as yet been proved that these very important local accumulations are connected with increase in the total methane volume issuing from the mine or with any of the seismic or meteorological conditions. The human factor is the controlling cause of most of the accidents. Many come from carelessness in the details of ventilation, or mismanagement of lamps or fire in other form and from blown-out shots. Ordinarily there are fewer accidents during times of evident danger than when there appears to be less need of precaution and miners take more risk. With dust explosions the factors are somewhat the same and most of the catastrophes are due not to any special condition in the mine but to some careless act or mishap that would have yielded the same result at any other time.

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INDEX.

A.	Page.	B.	Page.
"A" mine, Ill., occurrence of gas in..... 200, 224 relation of depth to, figure showing.. 223 effect of cessation of mining on..... 219		"B" mine, Ill., gas pressure in, tests of.... 217, 218 occurrence of gas in..... 201, 224 relation of depth to, figure showing.. 223 effect of cessation of mining on..... 219	
Abbott bed, Pa., workings in, occurrence of gas in..... 123, 127, 154, 156, 157, 167 effect of cessation of mining on. 145, 146 relation of, to output of coal..... 124 to coal surface exposed..... 124 structure of..... 123, 124, 128		Baltimore bed, Pa., workings in, area of..... 131 occurrence of gas in. 129, 131, 133, 153-162, 164 effect of cessation of mining on.. 145, 147 structure of..... 128 <i>See also</i> Cooper bed; Bennett bed.	
Abel, Frederick, on blower in Wallsend colliery, England..... 20 on relation of weather to gas emanation.. 82		Baltimore No. 5 mine, Pa., Baltimore workings in, occurrence of gas in..... 153 structural features of..... 153	
Abercarne mine, Wales, gas outburst in..... 22		Barker, Perry, on occurrence of gas in coal.. 45,	
Acieriers de France, mine at, analysis of gas in..... 19		46, 221	
Agrappe mine, Belgium, gas outbursts at, discussion of..... 22-24 pressure of gas in coal in..... 227		Barometric pressure. <i>See</i> Atmospheric pressure.	
Air pressure. <i>See</i> Atmospheric pressure.		Beard, J. T., on relation between earth movements and gas emanation..... 99	
Albert mine, Germany, analyses of mine air in..... 49 occurrence of gas in..... 50		Beaulieuart mine, Belgium, gas outbursts in, figure showing location of..... 67 results of..... 68	
Alfred bed, France, gas pressures in..... 70, 71		gas pressure in..... 227	
Altanwald mine, Germany, analyses of mine air in..... 49 occurrence of gas in..... 50		curves showing..... 68 tests of..... 67, 68	
Andros, S. O., air samples collected by..... 186, 189, 207		Bedson, P. P., on gases in English coal..... 33	
Anzin, France, mines at, analysis of gas in..... 19		Behrens, —, on effect of air pressure on gas emanation..... 88, 89	
Anthracite, escape of methane from..... 38		Belgium, coal beds in, gas outbursts in..... 22, 23	
occurrence of methane in..... 104, 105		gas pressure in..... 64-66	
Anthracite mines, gas explosions in..... 105		proportion of methane in..... 16	
pressure of gas in..... 170		<i>See also</i> various mines named.	
Arnould, M. G., on gas outbursts in coal.... 23, 35		Belle-Vue mine, Belgium, gas pressure in.. 64, 227	
Ath-Gouley mine, Germany, methane emanation in..... 59, 230		curves showing..... 66	
effect of atmospheric pressure on.... 84, 85		results of tests of..... 65, 66	
Atmospheric pressure, effect of, on gas emanation..... 82-98, 231, 232		curves showing..... 65	
at fresh coal surfaces..... 94, 95		Bellevue mine, Pa., workings in, methane emanation from..... 148	
curves showing..... 85, 87, 89		Bennett bed, Pa., workings in, analyses of air from..... 110, 151	
in old workings..... 92-94		map of..... 113	
Auchincloss mine, Pa., workings in, figure showing section through..... 158		occurrence of gas in..... 110,	
occurrence of gas in..... 157-159		113, 118, 133, 152, 153, 164	
structural features of..... 157, 158		effect of cessation of mining on.. 145	
Austria, coal mines in, air from, analyses of.. 51		relation of, to output of coal..... 120	
blower gas from, analysis of..... 17		to coal surface exposed..... 120	
outflow of gas from..... 230		structure of..... 128	
relation of air pressure to gas emanation in..... 90		<i>See also</i> Baltimore bed.	
<i>See also</i> various mines named.		Benton mine, Ill., gas from, analyses of..... 182	
Austrian Fire Damp Commission, investigations of..... 17, 53, 90		rate of, escape of..... 43, 44, 181	
Avondale mine, Pa., workings in, occurrence of gas in..... 169		curve showing..... 43	
		gas pressure in, tests of..... 216-218	
		occurrence of gas in..... 218	
		effect of cessation of mining on..... 219	
		relation of depths to, figure showing. 223	
		Bertrand, C. E., on differences in coals..... 28	
		Bessèges, France, gas outbursts at..... 22, 23	

	Page.		Page.
Bettina mine, Austria, analyses of air in.....	51	Cartersville mine, Ill., analysis of air from...	202, 203
Big Muddy mine, Ill., occurrence of gas in.....	196, 224	occurrence of gas in.....	194, 224
effect of cessation of mining on.....	219	relation of depth to, figure showing.....	223
relation of depth to, figure showing..	223	Castrop, Germany, mine at, analysis of gas in..	18
Bituminous coal, escape of gases from.....	32, 38	Cayuga mine, Pa., workings in, methane	
Blackbrier mine, Ill., gas pressure in, tests		emanation from.....	149
of.....	217, 218	Celynne mine, Wales, gas pressure in.....	64
occurrence of gas in.....	193, 224	Chamberlin, R. T., analyses of mine gas	
effect of cessation of mining on.....	219	by.....	16, 18, 19, 171
relation of depth to, figure showing..	223	on gas in coal.....	28, 29, 35, 36, 38, 40-42
Blair No. 1 mine, Ill., occurrence of gas in..	213, 224	results of investigations by.....	98
relation of depth to, figure showing..	223	Checker bed, Pa., workings in, air from, anal-	
Blanz, France, mine at, analysis of gas in....	19	ysis of.....	151
Bliss mine, Pa., workings in, gases from....	160, 161	volume of.....	137
"Blower," definition of.....	20	occurrence of gas in.....	137, 138, 152, 153
duration of.....	20	Chesneau, G., on methane emanation from	
origin of.....	20	coal.....	54, 91, 99
volume of gas given off by.....	20	Christopher, Ill., gas in mines near.....	183-184
<i>See also</i> Feeder.		Clark bed, Pa., workings in, gas from..	149, 150, 171
"Blower gas," composition of.....	15-17	Clifford mine, Ill., volume of methane from..	45, 46
proportion of methane in.....	14, 15	Coal, fresh, escape of gas from.....	47
Bochum, Germany, coals from, escape of gas		gas emanation from, factors affecting. 36, 225, 231	
from.....	47	lump, escape of gas from.....	37, 38
Bockwa-Hohndorf mine, Saxony, air in, com-		rate of.....	41
position of.....	53	permeability of, to gas.....	76
volume of.....	53	porosity of, effect of, on methane emana-	
occurrence of gas in.....	58	tion.....	49
Boldon colliery, England, gas pressure in....	62	pulverized, escape of gas from.....	39
methane emanation from.....	60, 75	weathered, volume of gas from.....	47
Bonifacius mine, Germany, analysis of gas in..	18	Coal bed, depth of, relation of methane emana-	
Bore holes, drilling of, to determine gas in		tion to.....	222
coal.....	170, 171	pressure of gas in, method of testing... 213, 214	
Boudouard, O., on absorption of oxygen by		tube for measuring, figure showing..	214
coal.....	81	strata overlying, permeability of, effect	
Bowkley bed, Pa., workings in, structure of..	128	of, on methane emanation.....	222
Brisbin mine, Pa., workings in, gas from....	149	Colp mine, Ill., occurrence of gas in.....	195, 224
Broockman, K., on effect of air pressure on		relation of depth to, figure showing.....	223
gas emanation.....	89	Connellsville, Pa., mine near, escape of gas	
on gases in German coal.....	34	from.....	43, 44
Brueckenberg mine, Saxony, air in, composi-		curve showing.....	43
tion of.....	53	gas outburst at.....	21
volume of.....	53	Consolidation mine, Germany, analyses of	
methane emanation from.....	58	gas in.....	18
Buddle, John, on gas outbursts.....	25	Cooper bed, Pa., workings in, analysis of air	
Bureau of Mines, investigations of.....	11, 175	from.....	110, 151
Burr C mine, Ill., occurrence of gas in.....	203, 224	map of.....	114
relation of depth to, figure showing..	223	occurrence of gas in.....	110,
Burrell, G. A., on accuracy in methane deter-		115, 118, 133, 152, 153, 164-167	
minations.....	103	effect of cessation of mining on. 145, 146	
on explosibility of fire damp.....	15	relation of, to output of coal.....	120
Bush mine, Ill., occurrence of gas in.....	206, 224	to coal surface exposed.....	120
relation of depth to, figure showing..	223	structure of.....	128
C.		<i>See also</i> Baltimore bed.	
C. & B. M. mine, Ill., occurrence of gas in....	204	Crushing, effect of, on escape of gas from	
Cadman, John, on ventilation of coal mines..	84	coal.....	36, 41, 42
Campagnac, France, mine at, analysis of gas		Culm, definition of.....	170
in.....	19	escape of gas from.....	169, 170
Cannel coal, gases in.....	32	utilization of.....	170
Carbon dioxide, characteristics of.....	13, 14	Cunyngham mine, workings in, occurrence	
rate of escape of.....	44, 47	of gas in.....	154
specific gravity of.....	13	D.	
Carbon monoxide, characteristics of.....	13	Darwin, G. H., on effect of air pressure on gas	
Carbondale formation, Ill., coal beds in.....	176	emanation.....	99
section of.....	176	Davis mine, Ill., occurrence of gas in.....	187, 224
		relation of depth to, figure showing..	223

	Page.		Page.
De la Beche, —, on inflammable gas in British mines.....	16	England, coal samples from, gases from... 32, 33, 34	
De Wolf, F. W., analyses of Harrisburg coal by.....	179	<i>See also</i> various mines named.	
on southern Illinois coal field.....	176	Eppleton colliery, England, gas pressure in..	62
Dechen mine, Germany, analysis of air in....	49	occurrence of gas in.....	75
Delaware mine, Pa., workings in, occurrence of gas in.....	152, 153	Essen, Germany, mine at, analysis of gas in..	18
structural features of.....	152	Ethane, escape of, from coal.....	47
Depth of coal, relation of, to volume of gas....	225	Explosions in mines, causes of.....	232
Dering No. 11 mine, Ill., gas from, composition of.....	215	F.	
gas pressure in, tests of.....	215, 216, 218	Faulting, effects of, on emanation of methane..	222
occurrence of gas in.....	188-190, 218, 224	"Feeder," composition of gas from.....	15, 16
effect of cessation of mining on.....	219	definition of.....	20
Dering No. 11 mines, relation of depth to, figure showing.....	223	<i>See also</i> Blower.	
Deutschland mine, Saxony, air in, composition of.....	53	Fire damp, composition of.....	14
volume of.....	53	explosions of, causes of.....	173
methane emanation in.....	58	fatalities from.....	173, 174
Diamond bed, Pa., workings in, occurrence of gas in.....	149	explosive, limits of, factors governing....	15
Diamond mine, workings in, occurrence of gas in (Pa.).....	148, 149	properties of.....	15
(Scotland).....	51	Five-foot bed, workings in, analyses of air from.....	110
Dickson mine, Pa., workings in, occurrence of gas in.....	148	map of.....	115
Dodson mine, Pa., workings in, occurrence of gas in.....	168	occurrence of gas in.....	110, 116, 129, 132, 133, 152, 153
Donetz basin, Russia, mines in, composition of air from.....	51	effect of cessation of mining on... 145, 146, 147	
Dorrance mine, Pa., workings in, analysis of coal from.....	131	relation of, to output of coal.....	120
analysis of gas from.....	19, 171	to coal surface exposed.....	120
map showing.....	130	structure of.....	128
occurrence of gas in.....	127, 128, 129, 131, 132, 133, 134	Floor, settling of, effect of, on gas emanation..	95
effect of cessation of mining on... 145-147		Fontenille, tests of French coal by.....	34
structure of.....	127, 128	Force, H. J., mine-air samples collected by... 158, 161, 169	
Dortmund, Germany, mines at, analysis of gas in.....	18	Forge bed, Pa., workings in, gases from.... 158, 160-162	
Dudweiler mine, Germany, analyses of mine air in.....	49	France, coal mines in, outflow of gas from... 16, 230	
Dufrane, A., on gas outbursts in coal mines ..	23, 24	<i>See also</i> various mines named.	
Dunmore bed, Pa., workings in, analysis of coal from.....	142	Frederic bed, France, gas pressures in.....	70
collection of air samples from.....	139, 140	Friedenshoffnung mine, Germany, occurrence of gas in.....	50
map of.....	140	Friedricksthal mine, Germany, analyses of mine air in.....	49
occurrence of gas in.....	141, 143, 148, 149, 150, 171	G.	
effect of cessation of mining on... 145, 147		Gabrielle mine, Austria, analyses of air in....	51
relation of, to coal surface exposed... 143		methane emanation in, effect of atmospheric pressure on.....	86, 87
structure of.....	141	variations in.....	88
Dunraven mine, South Wales, composition of blower gas in.....	17	Galatia mine, Ill., occurrence of gas in.....	207, 224
Duquoin, Ill., gas in mines near.....	185-187	relation of depth to, figure showing..	223
E.		Galloway, William, on barometric changes in coal mines.....	83, 84
Earthquakes, relation of, to gas emanation .. 98, 99		Garwood mine, England, blower in.....	20, 21
curves showing.....	100, 101	Gas in coal, composition of.....	225, 226
East mine, Ill. <i>See</i> Standard mine, Ill.		variation in.....	226
Eldorado, Ill., mines near, analysis of air from.....	210, 211	emanation of, effect of removal of coal on	95
Eleven-foot bed, Pa., workings in, gas from... 163, 172		effect of temperature on.....	97, 98
		factors governing.....	19, 20, 226
		relation of, to atmospheric pressure..	82
		curves showing.....	85, 87
		to earth tremors, curves showing.....	100, 101
		to variations in air pressure....	231, 232
		variations in conditions determining	97
		volume of, conditions governing....	225, 228, 229
		determination of.....	73-75

	Page.		Page.
Gas in coal, volume of, relation of, to temperature.....	80	Harry E. mine, Pa., workings in, occurrence of gas in.....	163
variations in.....	12, 228	Harton colliery, England, gas pressure in.....	62
Gas pressure in coal beds, effect of rock pressure on.....	75, 76	Hart-Williams mine, Ill., analyses of gas from.....	222
effect of water pressure on.....	75, 76	gas pressure in, tests of.....	215, 218
measurement of, tube for, figure showing.....	214	occurrence of gas in.....	180, 218, 220, 221, 224
method of testing.....	213, 214	relation of depth to, figure showing.....	223
variation of.....	60, 61, 227, 228	Hebburn colliery, England, analyses of gas in.....	16
causes of.....	63	Heinitz mine, Germany, analyses of mine air in.....	49
figure showing.....	63	methane emanation from.....	50
<i>See also</i> various mines and coal beds named.		Helse, —, on effect of air pressure on gas emanation.....	89
Gaseous mine, definition of.....	48	Hemlock mine, Ill., occurrence of gas in.....	197, 198, 224
Gateshead colliery, England, analyses of gas in.....	16	effect of cessation of mining on.....	219
Gemeinschaft mine, Germany, outflow of gas from.....	54, 59, 230	relation of depth to, figure showing.....	223
effect of atmospheric pressure in.....	84, 85	Herbst, —, on effect of air pressure on gas emanation.....	89
variations in.....	59	Herin mine, France, volume of methane from.....	54, 91
George bed, Pa., mine workings in, gases from.....	162	Herne, Germany, mine at, analysis of gas in.....	18
Georges Creek district, Md., escape of air through coal bed at.....	77	Herrin, Ill., mines near, analyses of air from.....	197-202, 205
Germany, coal samples from, determination of gases in.....	33, 34	Herrin bed, Ill., analyses of coal from.....	178
<i>See also</i> various mines and beds named.		features of.....	176
Ghyzen, Henri, on gas pressure in coal.....	66, 67, 69, 227	workings in, occurrence of gas in.....	180-207
on tests of French coal.....	34	Hetton colliery, England, gas pressure in.....	62
Glückhills colliery, Austria, analysis of gas in.....	17	Hillman bed, Pa., workings in, analyses of air from.....	110, 151
Gouley-Gemeinschaft-Königsgrube mine, Germany, volume of methane from.....	50	map showing.....	117
Grassy Island mine, Pa., workings in, gas from.....	149, 150	occurrence of gas in.....	110, 117, 118, 123, 125, 126, 129, 133, 154-158, 161, 162, 164, 168, 171
Gray, Samuel, on gas outburst at Otto colliery, Pa.....	21	effect of cessation of mining on.....	145-147
Gray, Thomas, analyses of air in Scotch mines.....	51	relation of, to output of coal.....	126
Green Ridge mine, Pa., workings in, methane emanation from.....	148	to coal surface exposed.....	120, 126
		structural features of.....	125, 126
H.		Hilt, C., on atmospheric pressure in coal mines.....	84, 85
H. Larisch mine, Austria, analyses of air in.....	51	on variations in methane emanation.....	54, 58, 59
Hainichen mine, Saxony, air in, composition of.....	53	Holden mine, Pa., occurrence of gas in.....	171
volume of.....	53	Hollenback mine, Pa., occurrence of gas in.....	155, 156
occurrence of gas in.....	58	structural features of.....	154, 155
Haldane, John, on oxidation of coal.....	80, 81	Höngen, Germany, mine at, analysis of gas in.....	18
on properties of methane.....	13	Horn mine, Ill., occurrence of gas in.....	187, 188
Hallidayboro mine, Ill., occurrence of gas in.....	206, 207, 224	relation of depth to, figure showing.....	223
relation of depth to, figure showing.....	223	Hoyt mine, Pa., workings in, air current in.....	136, 137
Hanna, Wyo., coal from, escape of gas from, curve showing.....	43	map of.....	135
rate of.....	44	occurrence of gas in.....	136, 137
Hannaford mine, Ill., occurrence of gas in.....	182, 224	effect of cessation of mining on.....	145, 147
relation of depth to, figure showing.....	223	structural features of.....	134
Hanover mine, Germany, effect of air pressure in.....	89, 90	I.	
Harrisburg, Ill., coal from, escape of gas from.....	43, 44	Ichon, —, on gas outbursts at Besaëges, France.....	22, 23
curve showing.....	43	Illinois, coal beds of, extent of.....	175
Harrisburg bed, Ill., analysis of coal from.....	179	structure of.....	175
features of.....	177	Illinois, southern, coal beds of.....	176
occurrence of gas in.....	207-212	structure of.....	175, 176, 177, 178
Harrison mine, Ill., occurrence of gas in.....	212, 224	map of.....	178
relation of depth to, figure showing.....	223	mines in, occurrence of gas in.....	179, 180
		<i>See also</i> various mines named.	
		Illinois State Geological Survey, cooperative agreement with.....	175
		Inkerman, Pa., mine workings near, gas from.....	150
		J.	
		Jarrow colliery, England, analyses of gas in.....	16
		gas outburst at, figure showing.....	25
		Johnston City, Pa., gas in mines near.....	191-194

K.		Page.			Page.
Karwin, Austria, mine at, proportion of methane in.....		17	Mahanoy City, Pa., mine near, gas outburst at		21
Karwin-Ostrau district, Austria, mines in, composition of air from.....		51	Majestic mine, Ill., occurrence of gas in....	187, 224	
gas pressures in.....		73	relation of depth to, figure showing..		223
Kidney bed, Pa., mine workings in, analysis of air from.....		151	Mallard, E., on atmospheric pressure in coal mines.....		86
occurrence of gas in....	129, 154-157, 164, 167		on gas pressures in coal mines.....	63, 71, 228	
Kingston No. 4 mine, Pa., occurrence of gas in.....		165, 166	Mansfield mine, Pa., coal from, escape of gas from.....		38
Knickerbocker mine, Pa., gas outburst at....		21	Manville mine, Pa., gas from workings in....		149
Koehler, G., on effect of air pressure in mines.	86-88		Maquet, A., on gas pressure in coal mines....		62,
Kohlscheid mines, Austria, gas emanation in, effect of air pressure in.....		88		64, 67, 69	
Konig mine, Germany, analysis of gas in....		18	Marchienne mine, Belgium, methane in.....		29
Kotsowsky, N., on analyses of mine air in Russian mines.....		51	Marcinelle-Nord mine, Belgium, gas outburst in.....		24
Kreuzgraben mine, Germany, analysis of gas in		18	pressure of gas in.....		61
L.			tests of coal from.....		34, 35
Lance mine, Pa., coal beds at, figure showing section of.....		108	Marcy bed, Pa., gas from workings in.....		150
structure of.....	108, 109		Maria mine, Höngen, Germany, analysis of gas in.....		18
thickness of.....	109		Marion, Ill., mines near, analyses of air from.	194-197	
cross section of, figure showing.....	109		"Marsh gas," origin of.....		12
river deposits above, map showing.....	121		See also Methane.		
gas from, effect of river deposits on.	121, 122		Marsilly, —, on volume of gas from coal.....		35
pressure of.....	170, 171		Marvine mine, Pa., occurrence of gas in.....		148
maps of.....	112-115, 117		Maxwell mine, Pa., workings in, gas pressure in.....		170
occurrence of gas in.	110, 115-120, 165, 166, 229		occurrence of gas in.....	156, 157	
effect of cessation of mining on.	145, 146		Mayer, Joh., on relation between atmospheric pressure and gas emanation.....		86
relation of, to output of coal.....	120		Meachem, F. G., on oxidation of coal.....	80, 81	
to coal surface exposed.....	120		Merthyr Vale mine, Wales, gas pressure in....		64
return air from, analyses of.....	110		Metamorphism of coal, effect of, on class of coal.....		28
Lange, C., experiments by.....	102		on methane content of coal.....	27, 28	
Langendreer, Germany, mine at, analysis of gas in.....		18	Methane, characteristics of.....		13
Le Chatelier, on effect of air pressure on gas emanation.....	86, 90, 91		composition of.....		13
on gases in coal.....	16		determination of, probable error in.....		103
Le Grande-Combe, France, mines at, analysis of gas in.....		19	emanation of, from coal.....		47
Lebanon mine, Ill., volume of methane from		45	curves showing.....		40
Lecocq, tests of French coal by.....		34	duration of.....		101
Lemaire, E., on gas outbursts in Belgian mines.....	23, 24		effect of cessation of mining on.....	58,	
Leonard bed, Liévin mine, pressure of gas in.	71, 72			59, 145-147	
Liévin mine, France, gas in, sources of.....	78, 79		effect of porosity of coal on.....		49
gas pressure in.....	69-72, 74, 227		effect of river deposits on.....		121
occurrence of gas in.....	56, 96		rate of.....		44
curves showing.....	93		relation of, to coal area exposed....	54, 56, 57	
relation of air pressure to.....	91, 92		to depth of coal bed.....		222
Liveing, tests of, to determine methane emanation.....		60	to kind of mining.....		55
Lombard, —, on gas outbursts at Bessèges, France.....	22, 23		variation in.....		229
Lothringen mine, Germany, analysis of gas in		18	conditions governing.....		60
Lugan mine, Saxony, composition of mine air in.....		53	from culm.....	169, 170	
methane emanation in.....	53, 58		in coal, amount of, relation of, to character of coal.....		102
Luzerne, Pa., mine gas from hole near, analysis of.....		18	condition of.....	28, 29	
M.			diffusion of, conditions governing....		31
McConnell, W., on gases in English coal.....	33		effect of metamorphism on.....	27, 28	
McLeansboro formation, Illinois, structure of.	176		origin of.....	26, 27, 231	
Madison mines, Ill., occurrence of gas in....	203, 224		processes in formation of.....	26, 27	
relation of depth to, figure showing..	223		volume of, to coal surface exposed....		55
			in return air, proportion of, conditions governing.....		48
			proportion of, to output of coal.....		52
			relation of, to age of coal.....		29
			to depth of coal.....		29
			to metamorphism.....	27, 28	
			to structure of coal.....		30
			to tightness of cover.....	29, 31	

	Page.		Page.
Methane, specific gravity of	13	O'Fallon mine, Ill., volume of methane from ..	45
See also various mines and coal beds named.		O'Gara mines, Ill., occurrence of gas in ...	207-211, 220, 221, 224
Mills bed, Pa., occurrence of gas in	158-162	relation of depth to, figure showing	223
Milne, J., on effect of earth movements on gas emanation	99	Orchard bed, Pa., workings in, gas from ..	165, 166, 168
Mine air, samples of, method of collecting ..	107, 108	Otto colliery, Pa., gas outburst at	21
See also various mines named.		Outbursts, gas, causes of	20-22, 25
Mine workings, size of, relation of, to volume of gas	225	effect of, on methane emanation	48
See also various mines and beds named.		Oxidation of coal, effect of, on methane emanation	46, 47
Mining, cessation of, effect of, on methane emanation	145-147, 219	result of	80
Monceau-Bayemont mine, Belgium, methane in	29	Oxygen, absorption of, by coal	81
Monongah, W. Va., coal from, escape of gas from, curve showing	43	escape of, from coal	47
rate of	44	proportion of, in mine gases	14
volume of	36-39, 43	Ovitz, F. K., on absorption of oxygen by coal ..	81
Morin, León, on methane emanation in Liévin mine	56, 57, 78, 79	on escape of gas from coal	42, 220
on percentage of methane in lump coal ..	39		
on relation between air pressure and gas emanation	91, 92, 95	P.	
tests of, on gas pressures in coal	71, 72, 227	Paintsville, Ky., coal from, escape of gas from ..	43, 44
Morrissey, B. C., gas outburst in mine at ..	22	Pancoast mine, Pa., occurrence of air in ...	148, 171
Murphysboro bed, Ill., coal from, analyses of ..	179	Paradise mine, Ill., analysis of air in	186
features of	177	occurrence of gas in	186, 224
occurrence of gas in	212, 213	effect of cessation of mining on	219
volume of gas in	212, 213	relation of depth to, figure showing ..	223
N.		structural features of	186
Nanticoke, Pa., escape of gas from strata near ..	77	Parr, S. W., analysis of Herrin coal by	178
Nanticoke mine, Pa., coal from, escape of gas from	36, 38	on escape of gas from coal	46
Naomi mine, Pa., coal from, escape of gas from	38	on gases in Illinois coals	45
Nasse, V. H., on air pressure in coal mines ..	84	Peabody mine, Ill., occurrence of gas in	45, 46, 196, 196, 224
Neu Iserlohn mine, Germany, analysis of gas in	18	relation of depth to, figure showing ..	223
occurrence of gas in	50	Pelham mine, England, blower in, gas from ..	20
Neunkirchen, Germany, diffusion of methane in mine at	31, 32	Pennsylvania, northern anthracite basin of, coal beds in, figure showing	106
New Virginia mine, Ill., occurrence of gas in	193, 194, 224	gases in	12
relation of depth to, figure showing ..	223	gas explosions in	105
Nigger Hill mine, Ill., occurrence of gas in ..	224	mines in, number of	105
relation of depth to, figure showing ..	223	volume of air in	105
Nitrogen, escape of, from coal	47	miners employed in	105
proportion of, in mine gases	14	production of coal in	106
Nottingham mine, Pa., occurrence of gas in ..	168, 169	structure of	106
No. 7 mine, Ill., occurrence of gas in	205	view of	106
effect of cessation of mining on	219	See also various mines and coal beds named.	
No. 8 mine, Herrin, Ill., occurrence of gas in ..	204	Petit, M. P., on gas pressures in coal	72, 73
No. 9 mine, Murphysboro, Ill., occurrence of gas in	212	Pettebone mine, Pa., structural features of ..	164
Oak Ridge mine, Ill., occurrence of gas in ...	224	occurrence of gas in	164
relation of depth to, figure showing ..	223	Pine Ridge mine, Pa., structural features of ..	150, 151
O.		workings in, analysis of air from	151
Oberhohndorf mine, Saxony, air in, composition of	53	Pittsburgh bed, Pa., workings in, gas from ..	172
volume of	53	Pittston, Pa., mine workings near, air current in	137
methane emanation in	58	analysis of coal from	139
Obernkirchen, Germany, coal bed at, analysis of gas from	18, 34	methane emanation from	137, 138
methane in	31	Pittston bed, Pa., workings in, analysis of coal from	139
		occurrence of gas in	137, 138, 163
		volume of air in	137
		Plat-de-Gier, France, mine at, analysis of gas in	19
		Playfair, Lyon, on inflammable gas in British mines	16
		Plymouth No. 2 mine, Pa., occurrence of gas in	167
		Pocahontas, Va., coal sample from, escape of gas from	43, 44
		curve showing	43